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A uranium resource evaluation of post-Proterozoic sedimentary formations in Alberta

by

© Christopher Wayne Van Dyke

A THESIS

SUBMITTED TO THE FACULTY OF GRADUATE STUDIES AND RESEARCH

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The undersigned certify that they have read, and recommend to the Faculty of Graduate Studies and Research, for acceptance, a thesis entitled A uranium resource evaluation of post–Proterozoic sedimentary formations in Alberta submitted by Christopher Wayne Van Dyke in partial fulfilment of the requirements for the degree of Master of Science.

ABSTRACT

The uranium resource potentials of Phanerozoic sedimentary formations in Alberta are evaluated by comparing their geology with those of the Powder River Basin, Wyoming and the Texas Coastal Plain in the U.S.A. A speculative resource of 3,000, 250,000 and 320,000 short tons U_3O_8 are indicated for the Upper Cretaceous Milk River Formation, Paskapoo Formation and Edmonton Group respectively. The Lower Cretaceous McMurray Formation has a speculative resource of 100,000 short tons U_3O_8. In terms of geographical distribution, the speculative uranium resources of the Oldman–Milk River, the Saskatchewan–Red Deer River, the Athabasca River and the Peace River drainage basins are 27,900 short tons U_3O_8, 360,000 short tons U_3O_8, 240,000 short tons U_3O_8 and 50,000 short tons U_3O_8 respectively.

The Upper Cretaceous Kneehills Tuff and its related bentonite horizons are the most extensive uranium source–rocks close to potential host formations. Precambrian shield terraines in northeastern Alberta and in adjacent areas of Saskatchewan are also significant uranium source areas. Potential uranium source–rocks further removed from favorable host–rocks include the Precambrian units in the Main Ranges of the Cordillera, the Mississippian Exshaw Formation, and Mississippian to Jurassic phosphate–bearing zones in the Rocky Mountains.

Coals in the Upper Cretaceous Whitemud Formation, the Battle Formation and the Scollard Member appear the most

favorable for hosting uranium. From Drumheller to Stettler the potential uranium content in coal, calculated from its calorific value, is moderately high. A regional radiometric anomaly occurring near Buffalo Lake supports this hypothesis.

Using petroleum industry gamma-ray logs to locate uraniferous subcrops was unsuccessful, due to the inability to differentiate anomalies produced by uranium-daughter isotopes (eg. ^{214}Bi) from those produced by ^{40}K and unknown variations of the drillhole configuration.

Fission-track analysis of Lexan plastic chips immersed in solution and irradiated in the SLOWPOKE reactor was proven too insensitive for uranium groundwater analysis in this region. Technical difficulties and a lengthly period for analysis prohibits any further development of this technique to obtain reliable groundwater uranium analysis in the required 1 to 10 ppb range.

Analytical procedures utilizing adsorption of uranium from groundwaters by activated charcoal were inaplicable, because the uranium background concentration in the charcoal was too high. The sample size was also too small for adequate detection using AECL neutron-activation analytical facilities.

Ultra-violet fluorescence analysis of groundwaters using the Scintrex UA-2 Uranium Analyser has a lower limit of detection of 0.08 ppb U. This sensitivity was sufficient to determine the distribution of uranyl-ion concentrations

in groundwaters of the Milk River Formation. Uranyl-ion contents were as high as 4.81 ppb and the uranyl-ion distribution pattern is similar to that of the sulfate-ion.

A computer study of the economics of mining sandstone uranium deposits in Alberta shows that at a possible contract selling price of \$40U.S./lb. U_3O_8 in 1990, deposits could be economically exploitable via open-pit mining, at ore tonnages as low as 0.5 million tons to 5 million tons with average ore grades of 0.10% U_3O_8 to 0.05% U_3O_8 if the deposit is near the surface (i.e. within approximately 100 feet). Underground mines would require a minimal average ore grade between 0.05% U_3O_8 and 0.10% U_3O_8 if the deposit tonnage is less than about 10 million tons.

ACKNOWLEDGEMENTS

Thanks are extended to the Alberta Government for their financial support of this project. For his supervision and criticisms Dr. R.D. Morton is gratefully acknowledged. The author would also like to thank the many people associated with the Alberta Research Council who provided access to files, and gamma-log records in addition to offering useful opinions and criticisms. Thanks also to Bob McDonald and Don Blackadar for their assistance in studying the gamma-logs. Equipment used for measuring the uranium content of groundwaters in the Milk River Formation was kindly loaned by Noranda Exploration Ltd., while the samples which were analyzed were provided by Dr. F.W. Schwartz. Lastly, and most importantly, thanks to to my parents, family and friends for their interest and support.

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I. Introduction

During the last decade it has become apparent that no nation, even resource-rich Canada, can afford to depend upon oil to meet its long-term energy requirements. In the short-term, continued economic growth in the 1980's can only be achieved by rapidly depleting existing conventional petroleum reserves. New oil, from frontier regions, enhanced-recovery techniques and non-conventional oil production from oil sands and shales can only constitute short-term solutions to meet our growing energy demands. During the next 50 years world energy supplies will have to be increasingly derived from other sources and from alternative fuels (see figure 1).

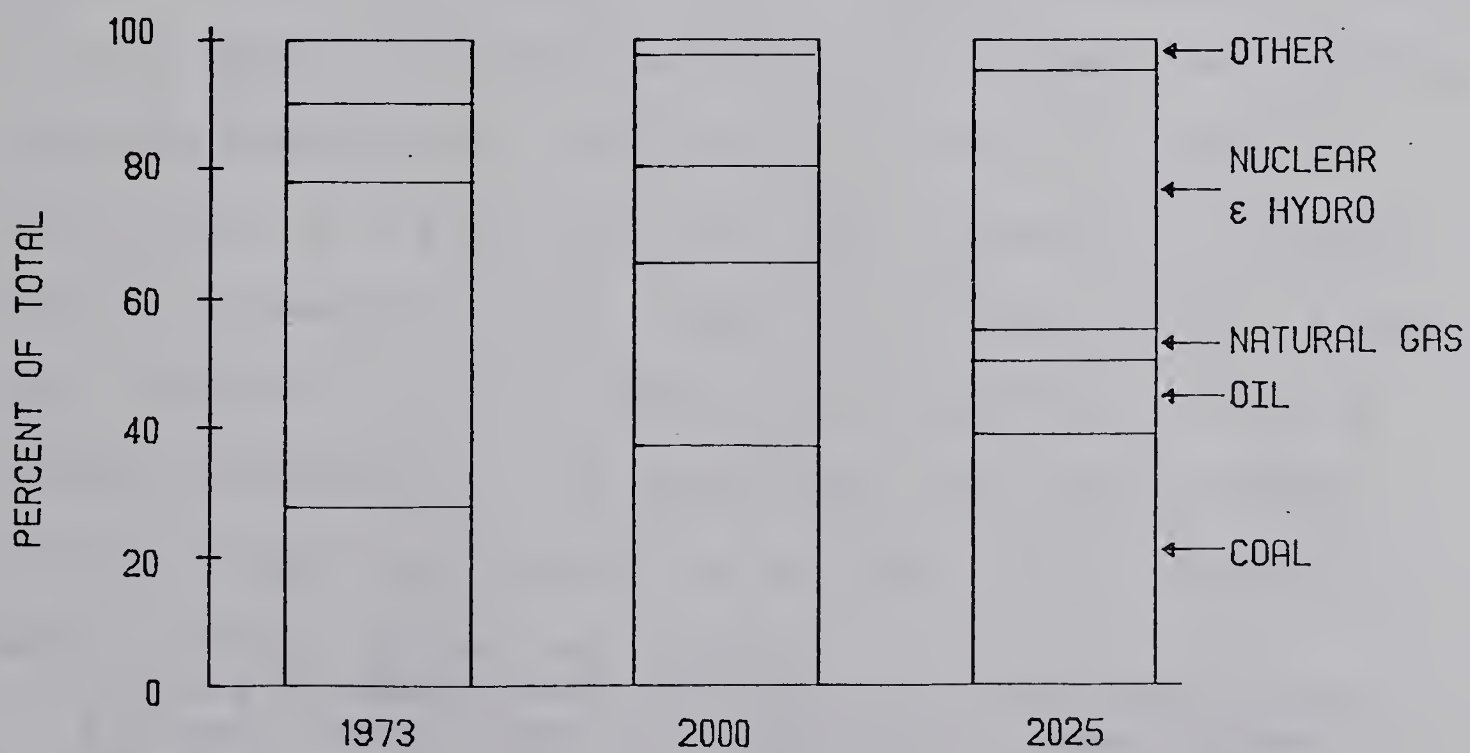

Fig. 1. Future world energy supplies (from Gander and Belaire, 1978).

Since 1973 the emergence of OPEC has caused many countries, including Canada, to realize that a policy of autonomy in energy supplies is in the national interest. To depend upon foreign sources of energy (paticularily oil) is intolerable for many countries because:

1. the cost of foreign oil is too great, resulting in slowed economic growth.
2. foreign countries cannot guarantee long-term deliveries, e.g. Iranian oil shipments have recently been interruped.
3. oil may be used as an agent of political blackmail and of subtle coercion.

Thus many short- and long-term incentives exist for any nation or province to formulate a policy for meeting its future energy requirements from a wide range of sources.

Canada has undertaken an energy assessment program to the year 2025 (Gander and Belaire, 1978). This study has made it clear that Canada will be unable to maintain the 5.3 per cent yearly growth-rate of energy consumption which has characterized the past two decades. Even at a reduced growth-rate of 2.8 per cent per year, Canada will have to double its present energy output by the year 2000. It has been forecast that electrical power output will have to increase four-fold by the year 2025, with the greatest increase being required before the year 2000. Gander and Belaire(1978) further state that:

> '...the nuclear power system is the one which offers the most ready scope for expansion to meet Canada's growing energy needs, at least for the next 30 years.'

Nuclear power production already compares favorably with other existing electricity-generating technologies from

an economic viewpoint. DeVoto(1978) states that an approximate cost-equivalence exists for producing electricity with $14/bbl oil, $29/ton coal, and $94-117/lb U_3O_8 ($244-304/kg U). * A further economic incentive for using nuclear energy in Canada is to support Canadian nuclear technology, particularly the CANDU reactor.

In terms of nuclear fuel supplies, Canada is presently in an enviable position. Federal government nuclear energy policy requires:

> '...sufficient uranium be reserved for domestic use to enable each nuclear power reactor currently on-stream, or planned to come on stream within the next ten years, to operate at an average annual capacity factor of 80% for 30 years from 1978, or from the in-service date of the nuclear unit, whichever is later. Further, domestic utilities are required to demonstrate to the Atomic Energy Control Board that they are maintaining a contracted 15-year forward supply for both operating and committed reactors.'

Even after meeting Canadian supply requirements and other existing agreements, Canadian uranium producers of the future will have an abundant supply of uranium for export. By 1985 only 15.3% of the estimated total uranium production of 12,500 tonnes scheduled will be for domestic consumption (Energy Mines and Resources Canada 1978).

However, if Canada hopes to maintain this level of production, new deposits must continue to be discovered during the next decade. Many deposits will either be exhausted or producing less uranium due to decreasing and less-economic ore grades. Unless major new deposits are

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*Values are given in Canadian dollars based upon $1.17Can=$1.00 US.

found, Canadian uranium production is forecast to decrease after 1990 (Energy Mines and Resources Canada 1978).

Alberta might be in a somewhat unique position with respect to nuclear fuel resources. During the early and mid-1970's the large upsurge of uranium exploration in Canada virtually ignored the potential for post-Proterozoic, sedimentary-hosted deposits in Alberta. On a global scale this type of deposit has historically been the most abundant source of uranium. Canadian exploration has traditionally been centred around Proterozoic uranium deposits within the domain of the Canadian Shield.

II. The Theory of Resource Evaluation

In the broadest sense, mineral-resource appraisals measure the degree of certainty of the geological availability of a given mineral commodity versus the feasibility of recovering the said commodity. Factors considered in this type of appraisal include: geology, economics, technology, time, fiscal climates, environmental regulatory climates and politics. None of these parameters is quantifiable in an exact manner and therein lies the great difficulty of "quantitative" resource evaluations. For this reason, great care must be taken to define terminology, explain concepts of classification and to describe the uses and limitations of the results. A description of any methods or equations used to quantify the mineral resources of a region should be given.

For the purposes of this study the following definitions of "reserve" and "resource" apply:

RESERVE - the current supply of profitably mineable ore-bearing material which has been delineated in three dimensions.

RESOURCE - the total potentially recoverable supply of a mineral (including known- and unknown-deposits, whether they are presently economically or uneconomically recoverable).

Both of these definitions are generalized to satisfy a broad range of applications. The terms are further delineated by the classification scheme and the methodology adopted in appraising a mineral resource. For further discussions of the concepts of resources versus reserves the reader is

referred to the Canadian Institute of Mining and Metallurgy "Forum" series of 1977 and 1978; Walrond and Morton (1977 and 1978).

A. Classification Schemes

The term "resource" encompasses such a wide range of circumstances that a classification scheme is necessary to quantitatively appraise the resource; two such schemes are illustrated in figure 2. "Measured" and "inferred–resources" are restricted to known productive uranium districts, as extensions of deposits or as undiscovered deposits along recognized geologic trends.
Areas not previously productive with or without any known uranium occurences are classified as "speculative resources."

The economic potential is generally measured in terms of the cost of production or of the required commodity selling price and the potential resources classified as either "economic"– or "subeconomic". A discrepancy arises when conditions change to alter the environment in which a resource is economically recoverable. That is, if the economic – subeconomic scale remains constant, then the price or cost scale changes with time and vice versa. For this reason, the time period for which the data apply should be clearly stated. The cost scale employed in this study is a best effort to predict the economic conditions which might apply around 1990 AD (with the help of the appended PRICE2

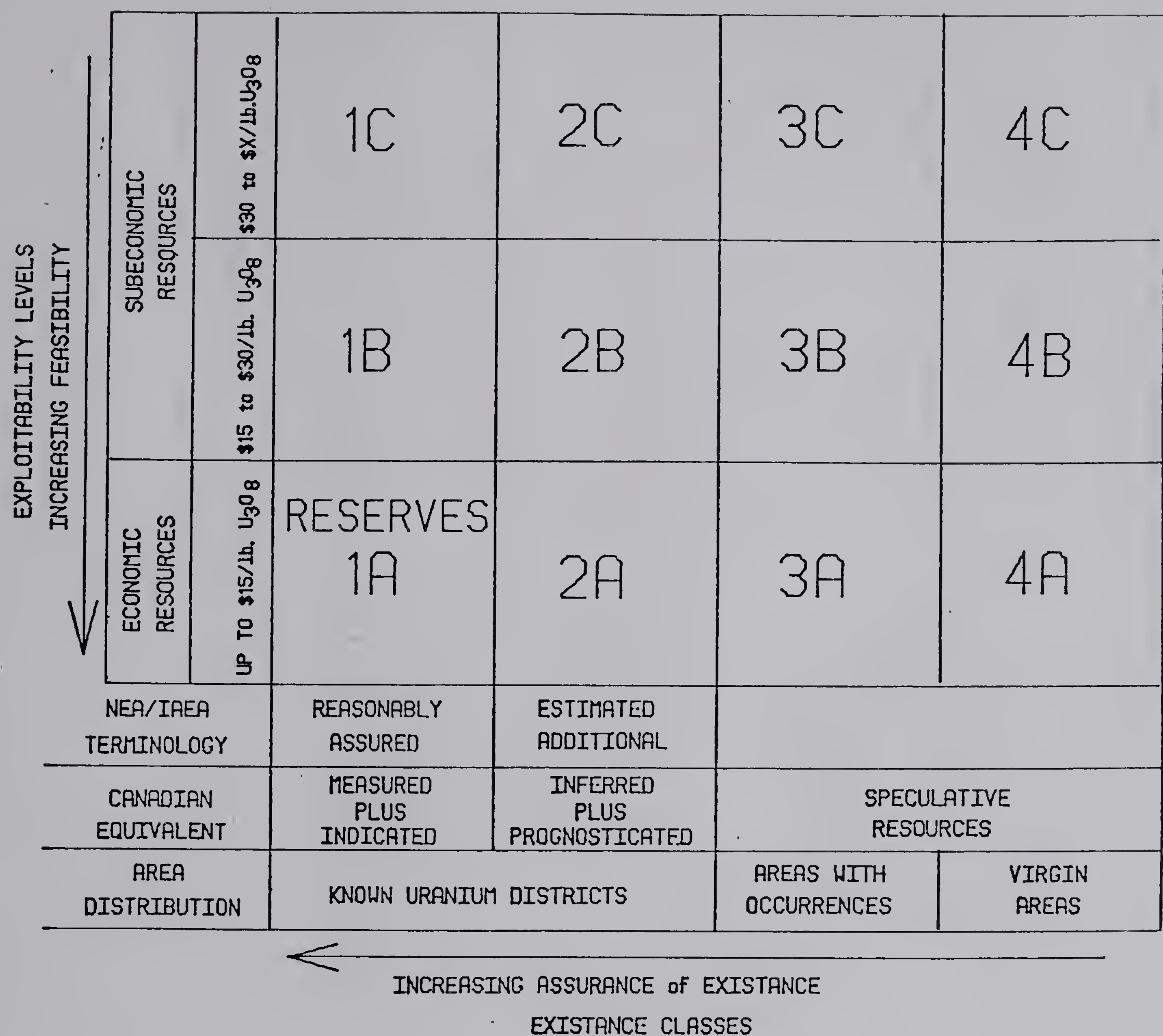

Fig. 2. Resource classification schemes (from Gander and Belaire, 1978).

computer program). This would be the earliest date by which any uranium deposit could likely go into production in Alberta.

The format of resource classification adopted in this study is a modified version of the Energy Mines and Resources Canada scheme (see figure 3). As no uraniferous deposits have yet been discovered in Alberta, the topics of this study fall under the category of truly speculative resources. The resource classification is economically based

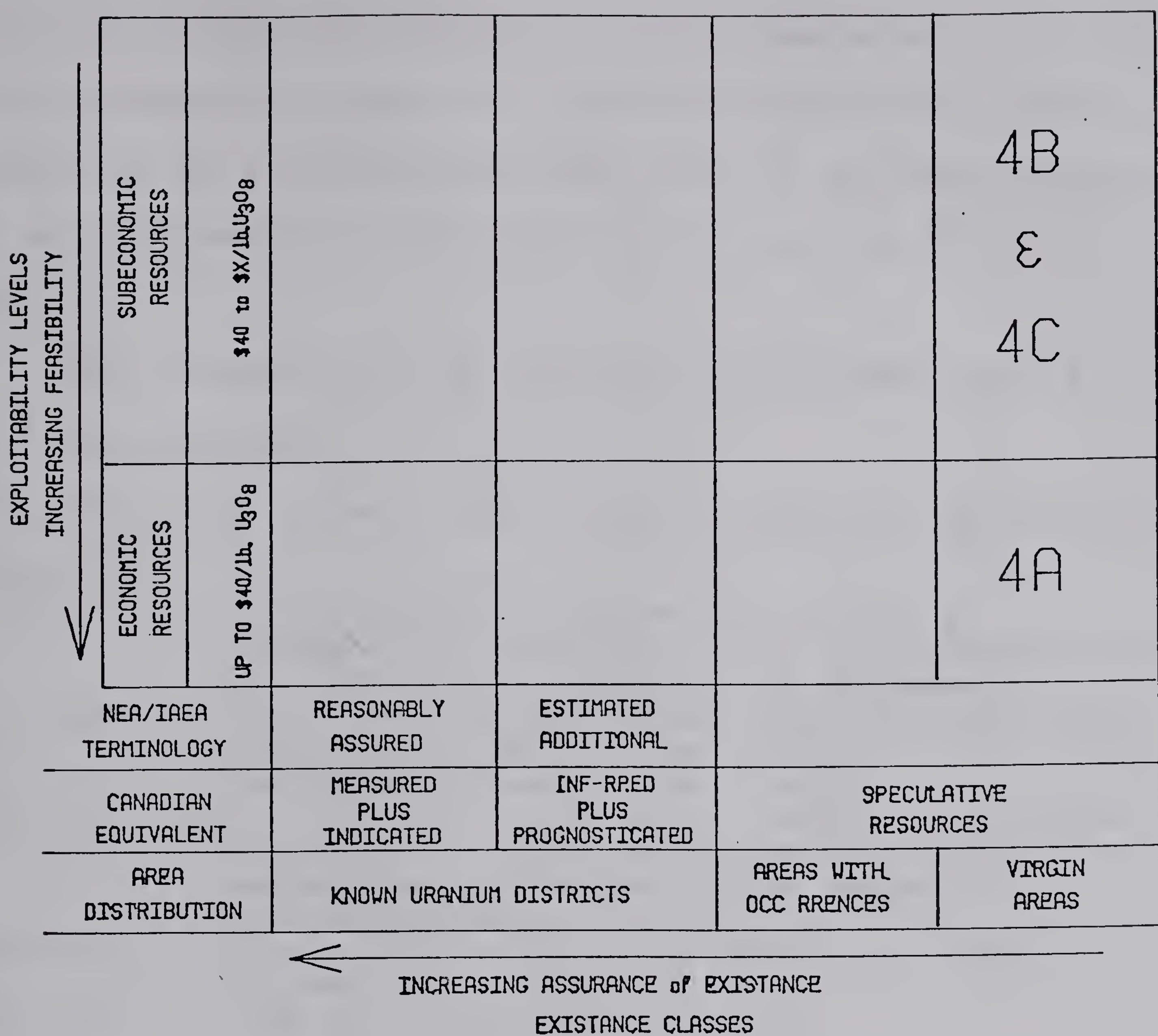

Fig. 3. Uranium deposit classification scheme applicable in Alberta.

upon whether the uranium oxide (yellowcake) is profitably saleable at a price less than or greater than $40 U.S./lb U_3O_8.*

B. Principles of Quantification

The underlying assumption of this study is that the potential uranium resources of an unknown area is approximately equal to that of a known uranium mining

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*See discussion of "PRICE2 program" and "Uranium Pricing" later in text.

district exhibiting similar geological characteristics. This study assesses the potential uranium resources of Alberta by comparing the Province's geology with the analogous geology of well-documented uranium mining districts in the United States.

Quantification of the resource is achieved using the following formula:

*Alta. res. = U.S. res. * min. ratio * area Alta./area U.S.*

where,

Alta. res. = the calculated uranium resource of a sedimentary formation in any given evaluation district within the Province of Alberta.

U.S. res. = is the sum of past production plus estimated resources of an uranium mining district within the United States of America.

Min. ratio = the mineralization ratio (which compares the geological similarity of the said Alberta sedimentary formation to the comparable U.S. formation or area).

Area Alta. = area of the sedimentary formation within the evaluation district.

Area U.S. = area of U.S. mining district.

Both the U.S. "resource" and "mineralization ratio" parameters are *subjective best estimates* lacking any statistical error estimation. The U.S. resource data are taken primarily from the United States National Uranium Resource Evaluation, Preliminary Report(1976). These figures include "measured", "inferred" and "speculative" resources mineable at a maximum cost of $30 U.S./lb. U_3O_8. Past production statistics were compiled from a number of sources, (primarily "Ore Deposits in the United States,1933—1967" and various "U.S. Bureau of Mines Yearly Reports" — see Table 1).

Table 1. Past production, reserves and resources of uranium from selected uranium mining camps in the United States.

District	Tons U_3O_8 Past Production	Reserves	Resources*
Powder River Basin	3,000	107,200	188,200
South Texas Coastal Plain	7,000	43,900	181,900

A wide range of geological characteristics are recognized as favorable indicators of uranium mineralization and deposit potentials. Numerical values are assigned to a list of these characteristics, whereby the most important indicators are ranked and given the largest values (see appendix 1). Where these factors are observed, in either the U.S. model area or in the Alberta evaluation district, the value assigned that characteristic is added to produce a quantitative measure of its uranium-hosting potential. The ratio of the Alberta total to the U.S. total is the so-called *mineralization ratio*.

"Resources" includes "reserves" estimate.

III. Methodology

A. Techniques and Strategies Adopted

From the table of mineralization factors (see Appendix 1) it is apparent that a wide range of data factors are available to assist the assessment of the uranium potential of a given area. An attempt is made in this study to investigate as wide a selection of these factors as available information sources permit. The following techniques have been utilized:

1. A review of potential uranium host-formations.
2. A review of potential uranium source-rocks.
3. A comparison of the characteristics of uranium-bearing coals in the United States with Alberta coals.
4. An evaluation of petroleum exploration gamma-ray logs and drill core.
5. An examination of groundwater reports and a limited pilot analysis of uranium in groundwaters.
6. A computer (PRICE2) study of economic considerations in underground and open-pit uranium mining.
7. A comparison of potential uranium host-formations in Alberta with uranium-producing formations in the United States.

The significance of each of these evaluation strategies varies from one region or formation to another, owing to their varying importance in forming or defining a uranium resource and due to the highly variable quality of data.

B. Format of data presentation

Discussion of Alberta's uranium potential in this paper is on the basis of six evaluation districts. These districts are defined by five major river drainage basins, covering the Plains, in addition to the Rocky Mountains and the

Foothills (see Figure 4). The decision to perform the study in this fashion was based upon the belief that it might at the same time facilitate the development of future land-use policies in terms of uranium exploration and environmentally controlled development.

In addition to the geographic descriptions of the Province's uranium potentials, data pertinent to the Province as a whole, or to some large sector of it are reported separately.

C. Review of Potential Host-Formations

An early phase of this study involved a review of published reports concerning the sedimentological characteristics of those strata underlying Alberta. Characteristics such as lithology, sedimentary depositional environment and sedimentary facies variations were investigated as they have long been recognized as key guidelines to the discovery of uranium deposits. On the basis of these types of information it is possible to identify:

1. Formations or regions which constitute possible uranium source-rocks.
2. Possible mechanisims of uranium mobilization in groundwaters, past and present.
3. Potential sites of uranium ore deposition.

In addition, the recognition of these characteristics is essential for determining which known U.S. uranium-producing districts one might utilize in forecasting, by analogy, the uranium potential of the areas under investigation.

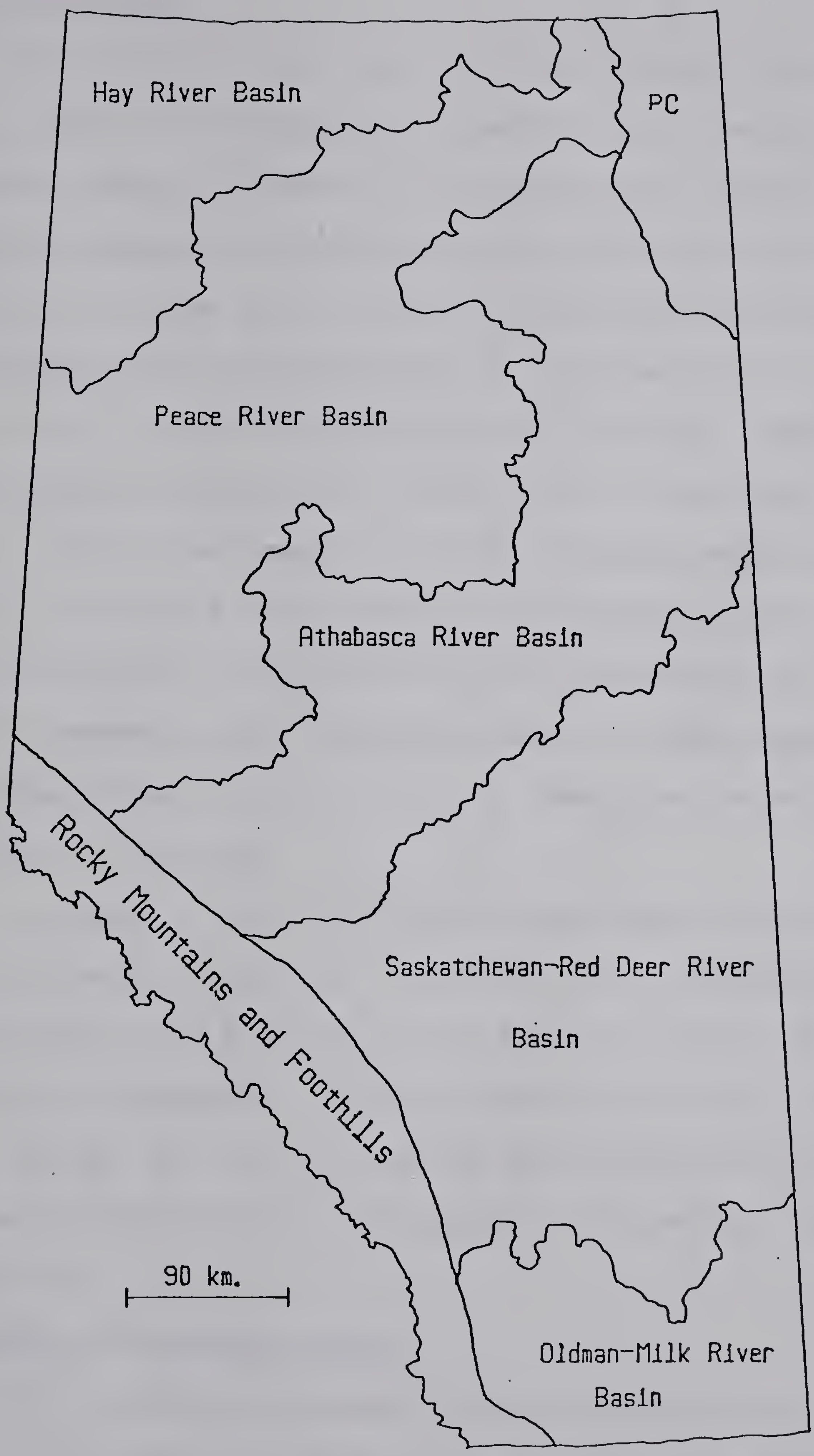

Fig. 4. Uranium evaluation districts in Alberta.

D. Source Rocks

Source rocks may herein be defined as: "formations which, due to lithology and location could constitute possible donors of uranium to groundwaters. This uranium might subsequently be deposited and concentrated to ore grades at another site." Those formations with relatively high-background concentrations of uranium (past or present) which are in close proximity to environments capable of concentrating uranium are clearly some of the most favorable source rocks. The location of any possible source-rocks is useful in ranking the potential of any nearby uranium host-lithologies and in postulating mechanisms of uranium deposit genesis which might be active in the region. Both exercises clearly would assist in designing an efficient exploration program.

A number of favorable lithologies have been identified in and around Alberta in strata of widely ranging ages. Often rocks from a given region and of similar age have similar lithologies, therefore source-rocks are reviewed in these terms. For clarity during the following discussions a series of correlative stratigraphic columns is given in Figure 5.

Precambrian Basement Suites

The Precambrian basement suites recognized as potential source-rocks for uranium in Alberta are of two types:

1. Medium- to high-grade metamorphic terrains of the Canadian Shield (exposed in northeastern of Alberta).
2. Low-grade metamorphic terrains in the Main Ranges of the

Rocky Mountains (found in western Alberta and British Columbia).

Both of these regions contain sectors with relatively high radioactivity backgrounds. This is not surprising, for in Canada, Proterozoic sediments and meta-sediments are recognized as the source-rocks of the major uranium-producing deposits in the N.W.T. and Saskatchewan.

The Canadian Shield and the Athabasca Sandstone

Airborne radiometric surveys recently undertaken by the Geological Survey of Canada have delineated a number of high-background radioactivity areas in close proximity to the major uranium mining camps in northern Saskatchewan (Richardson and Carson, 1977). Areas above the 1 ppm eU contour produced in the airborne gamma-ray spectrometry survey are considered areas of high concentration. Airborne measurements of 1-2 ppm equivalent uranium approximately equal 4-6 ppm eU in the underlying bedrock of the Canadian Shield (Richardson and Carson, 1977). One such region, near the Uranium City-Eldorado mining camp and which includes the Maurice Point discoveries of Uranerz Ltd. is immediately adjacent to the Alberta-Saskatchewan border. This region of high-background radioactivity probably extends into Northeastern Alberta's section of the Precambrian Shield. Godfrey and Plouffe (1978) have delineated a number of airborne- and minor ground-survey radiometric anomalies in the Alberta section of the Canadian Shield. Those anomalies from the Precambrian Shield (as opposed

to glacially deposited debris) are found primarily in high-grade metamorphic, granitoid terraines, with a lesser number of anomalies originating from metasediments. This characteristic is comparable to the pattern of high-background radioactivity observed by Richardson and Carson (1977) in northern Saskatchewan.

The Athabasca Group (and its underlying metamorphic basement) in Saskatchewan contains some of the largest and richest uranium deposits in the world. Sediments or groundwaters derived from any eroded uranium deposits on the western side of the basin could have supplied large volumes of uranium to subsequent sites of deposition, most likely in northeastern Alberta.

The potential of Post-Precambrian fluvial-deltaic sediments in northeastern Alberta as uranium exploration targets is enhanced by their close proximity to metamorphic terraines of the Canadian Shield and to the Athabasca Group. In particular, the Cretaceous McMurray Formation and the Grand Rapids Formation would appear highly promising, as their sediments were derived from these source-rocks and each is characterized by paleo- and modern- environments capable of concentrating uranium.

The Main Ranges of the Canadian Rockies

The Main Ranges of the Rocky Mountains have two regions which exhibit high-background concentrations of uranium, thereby characterising them as potential

source–rocks. During a truck–borne scintillometer survey of various roads in Central and Southern Alberta (this study) the average background radiation overlying the Purcell Supergroup was about 35 cps, whilst similar measurements in the Alberta Plains average 20–25 cps. In addition, Cu–U mineralization is said to have been found around Spionkop and Yarrow Creeks within the Grinnell Formation. Assays as high as 0.1% eU_3O_8 have been measured from grab samples from this area (E. Goble pers. comm.). The extent of the uranium mineralization is unknown but the copper mineralization is very extensive (Morton et al.,1974), suggesting that a source of uranium might have existed in the southern Rockies.

Further north, in the Jasper region (this study), a truck–borne radiometric survey detected gamma radiation levels of 120–140 cps within the lower Miette Group. Bell (1977) also detected radiation levels seven (7) times that of normal background levels. Further investigation is necessary to determine the full potential of the Rocky Mountains as a source terrain, because units of similar ages and lithologies are extensive in the Main Ranges of the Rocky Mountains.

Other than the Upper Tertiary Cypress Hills Formation and Hand Hills Formation, the Rocky Mountains were not the source of sediments for formations in the Alberta Plains or Foothills. Any uranium derived from the Rocky Mountain source–rocks could only be

transported by relatively recent groundwaters. Due to this fact, the only promising exploration targets, based upon the Rocky Mountains as a source of uranium, are those units in the foothills, immediately adjacent to the mountains (as is often the case with uranium deposits in the western United States) or Precambrian clastic formations of the Main Ranges.

Devonian and Mississippian Pelites

Organic-rich, black pelites should also be examined as potential source-rocks for uranium. On a global scale, black shales are enriched in uranium with a typical concentration range of 1.4 - 80 ppm U., compared with an average crustal concentration of 3.2 ppm U (Wedepohl, 1978). Their importance as uranium source-rocks is further enhanced by the fact that black shales commonly extend over large geographic areas with the potential of releasing a considerable amount of uranium to surrounding host environments to facilitate development of younger epigenetic deposits. For example, an estimated 5 million tons of uranium exists in the Chattanooga Shale in a twelve county region of central Tennessee.

Studies of uranium in pelites indicate that the uranium concentration is directly proportional to the organic content of the shale. The Chattanooga Shale in the United States and the Alum Shales of Sweden have average uranium concentrations of 0.006% and 0.02% U respectively. Both of these shales are anomously, enriched in organic-matter, with

concentrations of 20% and 30% respectively (see figure 6).

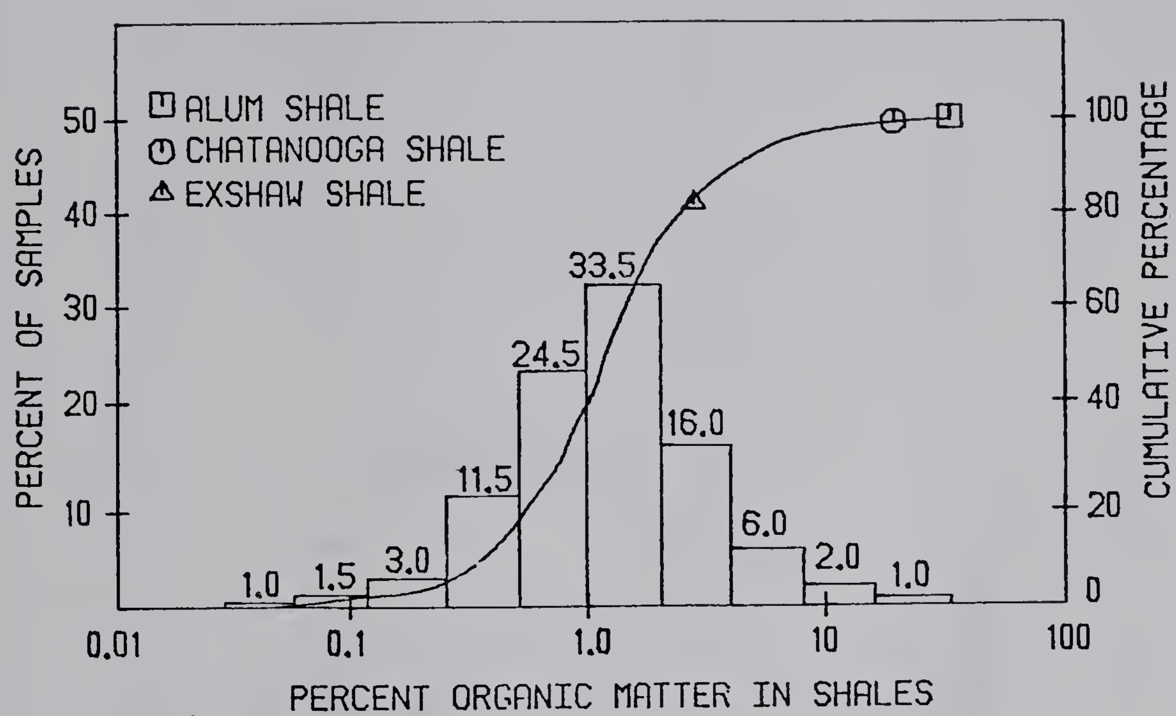

Fig. 6. Organic matter contents in shales (from Blatt et al., 1972 and Wedepohl, 1978). Where carbonized wood fragments comprise 99% of the Chattanooga Shale, the uranium is further enriched to a level of 0.7% U and in the Alum Shales, organic-rich nodules have uranium contents of 0.4% U. Thus any pelitic formation enriched in organic carbon warrants consideration as a possible uranium source-material.

Secondary features influencing the uranium content of the pelite are the phosphate and pyrite contents of the rock. Where each of these is enriched, the uranium content is enriched concomitantly.

In Alberta, Devonian and Mississippian shales are noteworthy as possible uranium source-rocks, particularly

the Exshaw and Banff Formations (see figure 7).

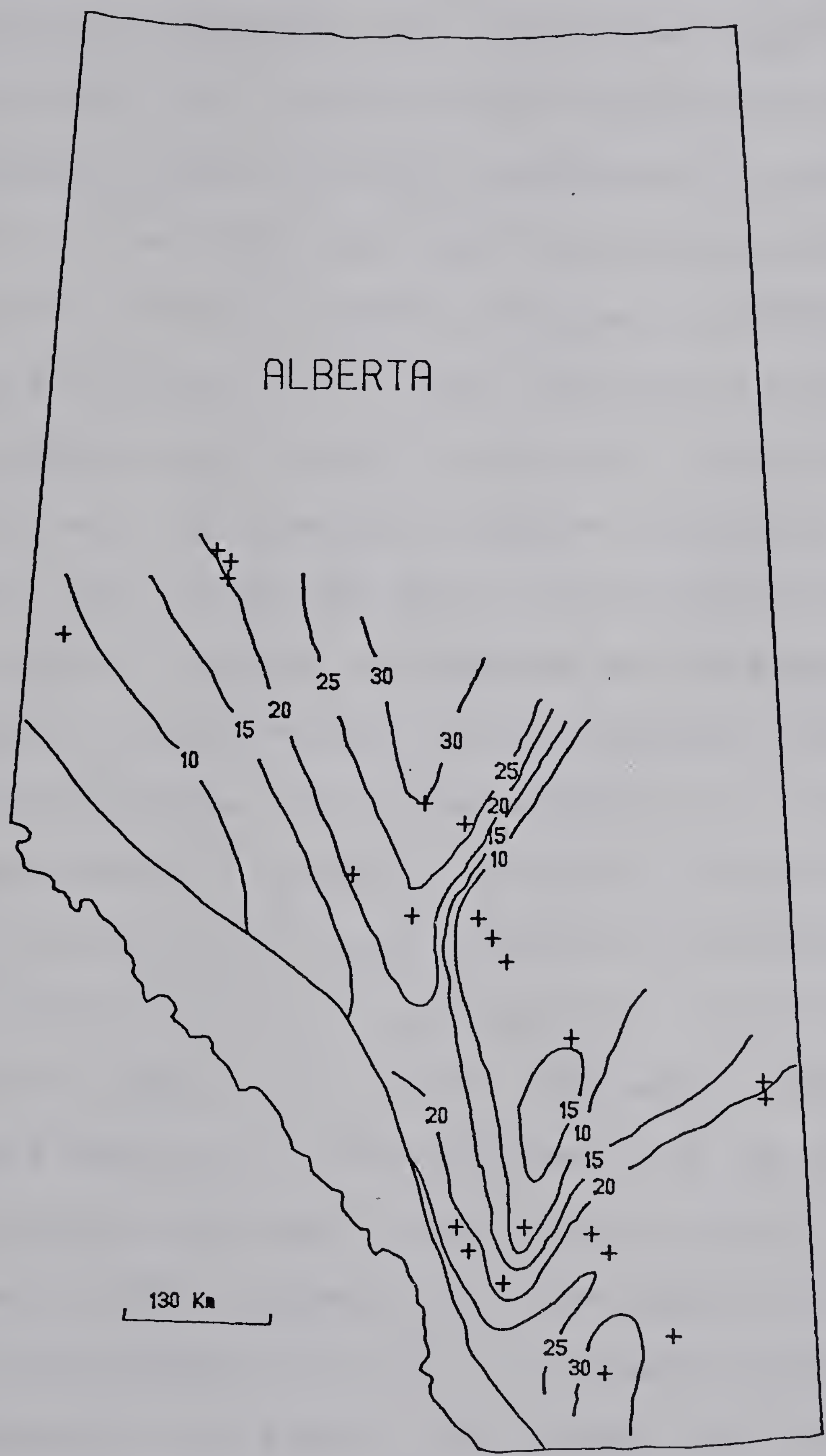

Fig. 7. Contour map of ppm. U in the basal four feet of the Exshaw Formation (from Campbell, 1980).

Both of these formations exhibit prominent gamma radiation anomalies on oil-well drilling logs.

In the Rocky Mountains of Alberta and the Foothills the Exshaw Formation consists of a lower black shale unit and an upper siltstone unit, while in the southeastern Plains, the Bakkan Formation (stratigraphic equivalent of the Exshaw Formation) has an additional black bituminous shale overlying the siltstone member. MacQueen and Sandberg (1970) postulate that the origin of the black shale unit of the Exshaw Formation is related to the event which developed other Devonian - Mississippian shales, including the uraniferous Chattanooga Shales in the United States. Campbell (1980) analyzed 155 samples of the Exshaw Formation from 22 drill core sections to measure their zinc, nickel and uranium contents. All three elements were progressively more concentrated in samples approaching the base of the formation, but only the uranium content was anomalously greater than the average concentration found in shales. The mean uranium concentration of the 155 samples was 10±2 ppm (1 standard deviation), while samples from the basal 4 feet of the formation averaged 20 ppm uranium (shown in figure 6). Campbell (1980) suggests that the uranium was concentrated syngenetically or during early diagenesis by organic matter in an euxinic environment, or that the metals were transported by hot brines from the underlying Wabamun Formation. The former hypothesis appears more probable as this is believed to be the method in which other organic

shales became uraniferous.

The main shortcoming of the Exshaw Formation as a source-rock is its remoteness from any favorable host lithology. Overlying units are predominantly marine limestones and dolomites of the Rundle Group, and underlying the Exshaw Formation are similar Devonian carbonates. Uranium is only rarely hosted within carbonates and mainly in karst environments or in ancient playas or sabkhas. Therefore, the only likely means of deriving uranium from the Exshaw Formation and transporting it to a suitable host-environment is by large-scale groundwater movement.

Only in the Rocky Mountains and Foothills is the Exshaw Formation sufficiently exposed or close to surface to have been active in recent hydrodynamic systems and thus available for uranium contribution. In this region, faulting could serve the dual purposes of bringing the source-rock closer to the host rock, and in possibly providing conduits for groundwater movement. As a source-rock only the western-most sector of the Exshaw Formation is therefore encouraging.

Mississippian to Jurassic phosphates

Phosphate-bearing pelites and phosphatic sandstones are commonly very large, low-grade sources of uranium. Where these phosphates are mined, uranium is often recovered as a by-product, and lower- grade phosphate horizons should be considered as a favorable uranium source-rock.

The U^{6+} ion is capable of limited substitution in carbonate fluorapatite in place of the Ca^{2+} ion. Additionally, uranium can be adsorbed onto apatite and concentrated in organometallic complexes in the organic fraction of phosphatic rocks.

Uranium concentrations in the Albertan phosphatic rocks have not yet been studied in detail, although all samples so far analysed (using gamma-ray spectrometry) range from 0.001 to 0.007% eU_3O_8. This lies within the range of concentrations commonly exhibited by uranium in marine phosphates. A general trend has been recognized in which the highest uranium assays are found within the richest phosphate deposits. Based upon this fact, uranium concentrations close to 0.007% U_3O_8 should be expected to be found in the richer phosphate deposits of Alberta.

Phosphorites and phosphatic pelites and sandstones have been mapped in the Rocky Mountains from the International Boundary through Alberta into Northern British Columbia. In the Mississippian through Jurassic succession, phosphates occur at four horizons as indicated in Table 2:

Early Mississippian phosphates are found in the Exshaw Formation from the Canada-U.S. border, northwards to Banff. Pelletal- and nodular-phosphate occurs in concentrations up to 23% P_2O_5 with uranium assays from 0.001 - 0.006% eU_3O_8. The thickness of the phosphorite layers in this unit is variable but averages about two feet in the most promising

Table 2. Names and ages of phosphatic horizons in Alberta.

Formation	Age
Exshaw Shale	Mississippian
Ishbel Group	Permian
Spray River Group	Triassic
Fernie Group	Jurassic

area between Crowsnest Pass and Race Horse Pass.*

The Ishbel Group (analagous to the Rocky Mountain Group name) phosphates are found in variable amounts in four of six formations. An Ishbel Group section is illustrated in Table 3 showing those formations which contain phosphate.

Table 3. Phosphate in the Ishbel Group.**

Formation	Description
Mowitch Formation	phosphate nodules and fish remains
Ranger Canyon Formation	phosphatic chert conglomerate and bone fragments
Ross Creek Formation	nodular and oolitic phosphate, and phosphatic siltstone
Telford Formation	–
Johnston Canyon Formation	phosphatic siltstone
Belcourt Formation	–

The lateral dimensions of the four phosphatic horizons are poorly documented. Phosphate concentrations in the Mowitch Formation and Johnston Canyon Formation seem to be little more than trace amounts and are likely unimportant.

It appears that significant phosphate mineralization in the Ross Creek and Ranger Canyon Formations is restricted to

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*Data compiled from Research Council of Alberta assessment files.

**Data collected from RCA assessment files and from McCrossen and Glaister et al., 1966.

southwestern British Columbia and to the Crowsnest Pass district of Alberta. Phosphatic beds up to 3.7 metres thick, and commonly 1 metre thick occur in these units. Grades as high as 27% P_2O_5 have been measured (Christie, 1979). Phosphates in these formations are the most northern extension of the Phosphoria Formation phosphates in the United States.

Triassic Spray River Group phosphates are observed throughout the Front Ranges of Alberta. A persistent 5 to 15 foot thick phosphatic shale and siltstone is found in the Black Shale Member of the Sulfur Mountain Formation. This unit continues into northern British Columbia where phosphatic pebbles, pellets, nodules and shales are found at the base of the Doig Formation. A strong gamma-emitting zone is often observed in the phosphatic zone at the base of the Doig Formation (McCrossen and Glaister 1966). The uranium concentrations in this phosphatic horizon have not yet been quantified.

The richest phosphates discovered to date in the Canadian Rockies are found in the basal section of the Fernie Formation, particularly in the Elk River-Crowsnest Pass region. In this area of British Columbia and Alberta, nodular and pelletal phosphate is found containing up to 35% P_2O_5 over a three foot thickness. Uranium contents range

0.001–0.01% eU_3O_8.z *

Low-grade phosphatic belemnitic beds are found in the lower Fernie Formation in southern Alberta and British Columbia. Uranium concentrations are within the range found in the basal Fernie phosphates, although the P_2O_5 concentrations and grades are lower.

<u>Upper Cretaceous bentonites and tuffs</u>

Due to its relatively large ionic radii (U^{4+} – 1.08A, and U^{6+} – 0.81A), and high +4 and +6 oxidation states, uranium is allowed a very limited substitution in common igneous rock-forming minerals. A low uranium concentration in magmas also prevents this element from crystallizing urano-silicate accessory minerals in most main stage, magmatic differentiates. Instead, uranium characteristically becomes enriched in late-stage melts and fluids, particularly those of calc-alkaline and felsic composition.

The volcanic equivalents of late-stage magmatic differentiates are of particular interest as a source of uranium. DeVoto(1978) notes that with volcanic rocks:

1. the uranium content is 1.5 – 2.0 times that of their equivalent plutonic rock.
2. uranium is readily leached from the surfaces of volcanic shards.
3. devitrification or dissolution releases uranium from the volcanic glass.

Late-stage, felsic, vitric pyroclastics are thus a potential source of uranium. DeVoto(1978) substantiates this point with the observation that the uranium concentrations in

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*Analysis done with hand specimens using a gamma-ray spectrometer.

groundwaters flowing through tuffs and tuffaceous sediments is usually relatively high, ranging from 20 to 200 ppb.

In addition, Adams and Weaver(1958) determined the uranium concentration of 69 bentonites collected from across the continental United States. Uranium concentrations in these bentonites ranged from 1.2 to 20.9 ppm.; with mean and median concentrations of 5.0 and 4.5 ppm. respectively. Late Cretaceous through Early Tertiary sediments in southern and central Alberta contain thin widespread, relatively continuous bentonite zones. Bentonite appears to be most important as a potential source-rock in the Edmonton Formation and the Paskapoo Formation, and their stratigraphic equivalents in south-central Alberta.

These altered tuffs could have been capable of releasing a considerable amount of uranium to circulating groundwaters. A typical bentonite zone one meter thick over an area of one square kilometer could release 250 tonnes of uranium to groundwaters if only 20 per cent of the uranium was released.*

E. Uranium in Coal

The association of uranium and carbonaceous (organic) material is a dominant characteristic of many sedimentary-hosted uranium deposits. A continuous spectrum of urano-organic deposits exists from sandstone-hosted

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*Assuming that the specific gravity of bentonite is 2.5 and the concentration of uranium in the bentonite is 5.0 ppm.

deposits with plant remains to uraniferous coals and shales. Generally uraniferous coals differ from the sandstone-hosted deposits in that:

1. Ore grades are generally lower.
2. Deposits are quite thin.
3. Ore mineralization extends over larger areas.

The organic matter in both types of deposits readily adsorbs or chelates any uranium from solution. For sandstone deposits, the carbonaceous material is generally isolated and limited in area. Uranium-bearing solutions are able to precipitate uranium over a long time period in a small area and thereby increase the ore grade. On the other hand, coal zones deplete groundwaters of their uranium over wide areas along the impermeable contact (if the coal is unfractured) between the coal and groundwater, forming thin, low-grade deposits (Vine, 1962).

These facts suggest that a slightly different approach to the recognizion of potential uranium-bearing coals would be useful. Delineating alteration zones and facies variations is a major technique for locating promising uranium-hosting zones in sandstones, but it is not applicable in prospecting for uraniferous coals. Low-grade coals are very extensive in Alberta and any method of differentiating between possible uranium- bearing coals and other low-rank coals would be highly desirable.

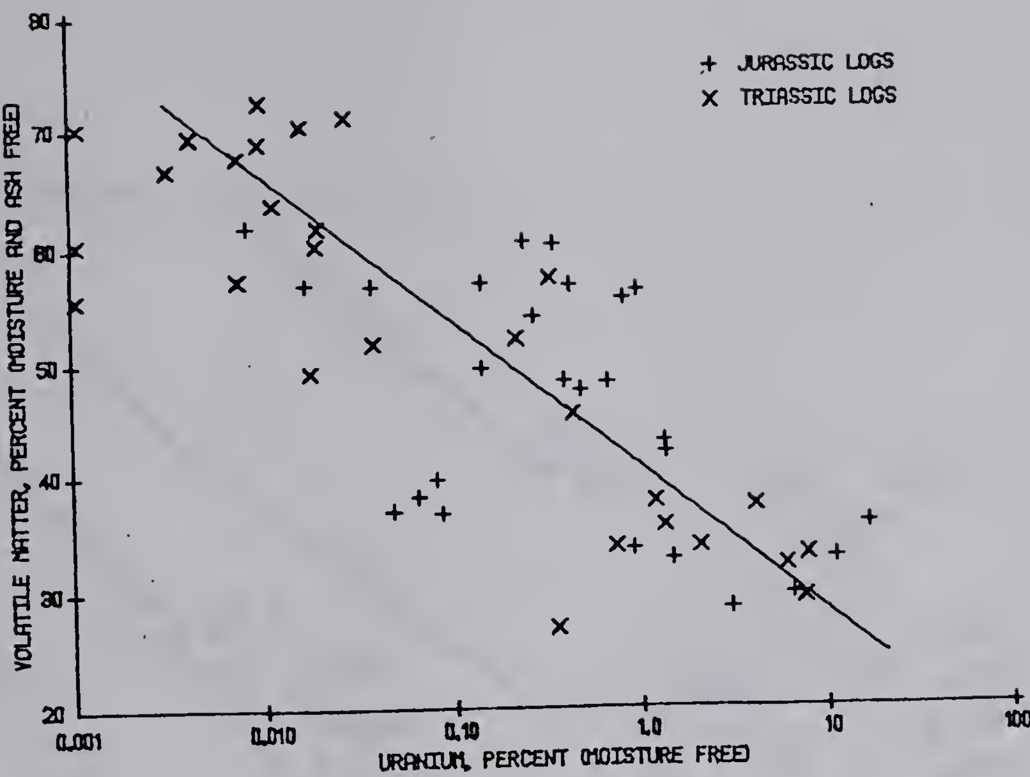
Breger (1974) analyzed 64 Triassic, Jurassic and Early Tertiary uraniferous coals and coalified logs from the Colorado Plateau and Wyoming to ascertain whether any

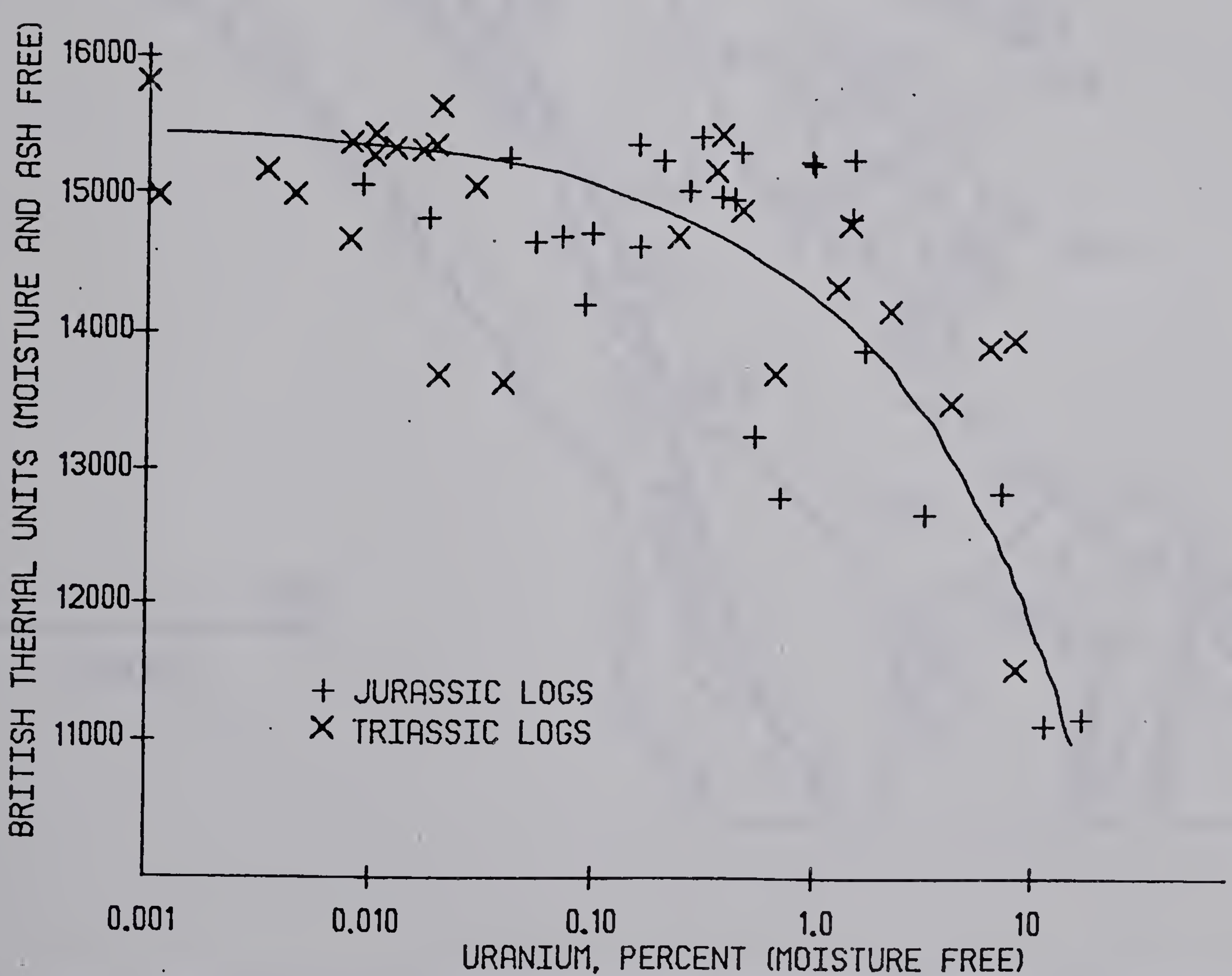
correlation exists between their uranium contents and commonly measured coal properties. It was determined that uranium significantly decreased with both the calorific value and with the volatile-matter content of the coal (see figures 8 and 9). Breger attributes this to the 'radiochemical dehydrogenation and demethanation of organic compounds' in the coal. Irradiation of coal was thought to result in the loss of both hydrogen and carbon. This present study attempts to identify coals in Alberta which appear to have undergone a similar decrease in their calorific value and their volatile-matter content using data published by Steiner et al. (1972). Only samples from the Alberta plains were considered, as all of Breger's (1974) samples were low-rank coals which have not undergone any dynamothermal metamorphism. It was assumed that any trends in either parameter which varied with the regional north-south coal-rank trend (parallel to the Rocky Mountains) and from which a high apparent "potential uranium content" was calculated, constitutes a "potential uranium anomaly". The results are shown in figures 10 and 11 (see appendix 2). These apparent uranium contents are not actual measurements of uranium concentrations but are merely predictions of the coal's uranium content based upon their calorific values and their volatile-matter concentrations assuming they follow the same trends determined in Breger's samples.

Two areas do show anomalously high apparent potential uranium concentrations calculated from their volatile-matter

contents. The smaller region lies just west of Edmonton, and the larger is in the Drumheller area. Coals from both localities were from the Edmonton Group, including the Scollard Member. The distribution of apparent uranium concentrations derived from the volatile-matter contents, closely parallels the trends shown by fixed carbon (moisture and ash free) values from Steiner et al. (1972). This trend in the fixed carbon content is apparently due to the regional change in rank and therefore it would be difficult to attribute any variation of the apparent uranium concentrations to the actual presence of uranium.

Potential uranium concentrations derived from the gross calorific values of Alberta coal yield only three single anomalous values, one each from the Paskapoo Formation, the Edmonton Group, and the Frenchman Formation. The only conclusion which may be drawn from the anomaly west of Edmonton (in the Paskapoo Formation) is that it confirms the suspicion that low-rank coals in this formation are favorable for hosting uranium. Due to the anomalous apparent potential concentration of the Frenchman Formation coal and the fact that isolated uranium occurrences with up to 825 ppm U have been found to occur in the Saskatchewan section of the Cypress Hills, the Frenchman Formation coals would seem to be quite favorable for hosting uranium.





Figs. 8 and 9. U contents(%) vs. the volatile matter content(btu/lb) and the calorific values of coal from the United States (from Breger, 1974).

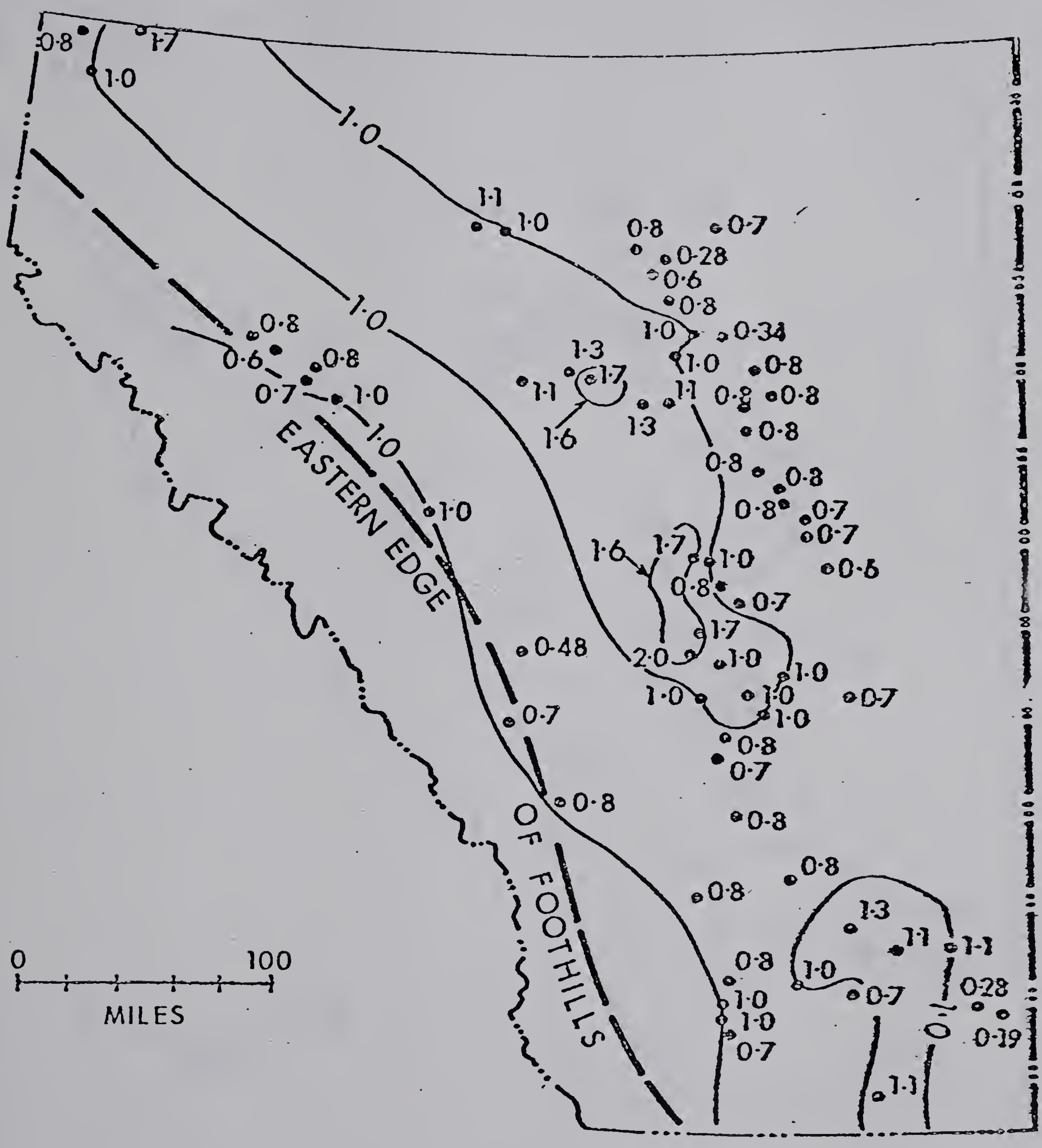


Fig. 10. Potential U content(%) of Alberta coals calculated from the volatile matter contents.

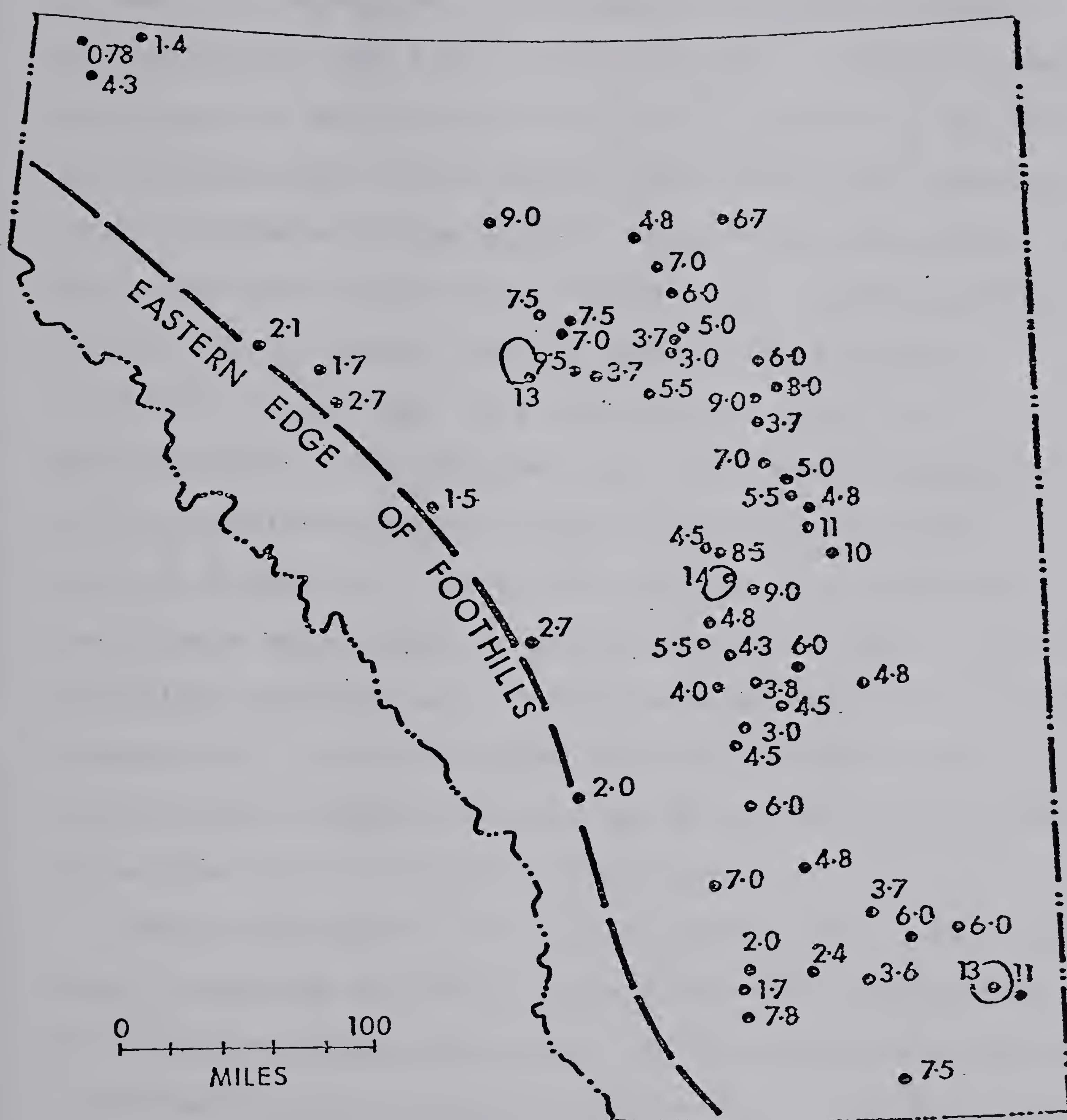


Fig. 11. Potential U content(%) of Alberta coals calculated from the dry calorific value.

A more noteworthy fact is that two areas underlain by the Edmonton Group, including the Scollard Member, exhibit, although not anomalous, high apparent potential uranium concentrations. The first, north and west of Edmonton, again parallels the variation of coal ranks in the area, yet the southernmost part of the region falls within the "potential" anomaly produced in the volatite matter–uranium correlation. The second area is in close proximity and slightly northeast of the volatile–matter–uranium anomaly found in the Drumheller region. The long axis of this region is perpendicular to the regional ranking of coal, suggesting that the variation might be correlated with potential uranium in the coal. The uranium content of the Sheerness coal from a single grab sample measures 0.69 ppm U, with an analytical precision of 4.15% (Mike Apps, pers. comm.). This suggests that the uranium contents predicted from the volatile matter concentrations may be more accurate as they range from 0.7 to 1.0 ppm U in this area.

Most coal found in the Alberta Plains lies within the Upper Cretaceous succession, except for thin lignite beds in the Tertiary Paskapoo Formation. In this succession the most predominant potential uranium source–rock is the Kneehills Tuff and related bentonite zones occurring in most of the Upper Cretaceous formations. Coal seams in the Upper Edmonton Group are closest to a potential source of uranium and would therefore be most likely to contain uranium mineralization. The important seams include the Ardley and

Nevis coal zones, which are within a couple hundred feet above the Kneehills Tuff (Holter et al. 1975), and the Carbon and Thompson coal seams which lie no more than 50 and 150 feet respectively below the tuff zone (Steiner et al., 1972).

The uranium-hosting potential of the Ardley and Nevis coal zones are diminished by the fact that groundwater reports concerning the near-surface Ardley coal zone trend show the regimes to be dominated by downward-moving groundwaters (Vanden Berg, 1969; Le Breton, 1971; Borneuf, 1972 and Tokarsky, 1977). This is indicated by flow directions from cross sections in these Alberta Research Council Reports. Where insufficient data exist to determine the flow direction, the trend of Ca-Mg bicarbonates changing to Na-bicarbonates was assumed to indicate the flow pattern. Additionally, the lithology of the intervening section contains both bentonitic zones and some thin coaly zones. The influence of these zones is unclear, but they could concievably capture uranium from groundwater. Gamma logs of the Scollard Member (Holter et al.,1973) exhibit no anomalously high peaks immediately below the Ardley coal zone, but do show moderatly high radiation in carbonaceous zones (coals and shales) farther below the coal zone. It would seem more likely that any uranium mineralization in the Ardley coal would be sporadic and of low grade. Intervening carbonaceous horizons along zones of high permeability would more likeky to host uranium than the

Ardley coal itself.

Owing to the fact that the Carbon and Thompson coal seams are below the Kneehills Tuff, with a small intervening section, which exhibits good groundwater permeabliity, they are considered to have the most favorable uranium host-potentials of all coals in Alberta.

F. Gamma-Ray Log Survey

Traditionally the gamma-ray log has been used by the petroleum industry to measure the shale content of a formation and as a stratigraphic correlation tool. Our study analysed published gamma-ray logs to determine their usefulness in identifying any formations which might exhibit high-background uranium contents. Such formations, or parts thereof, can either be favorable sources for, or hosts of, significant uranium mineralization. The potential advantages of this approach are that it utilizes a readily available database covering most of the Province and provides some measurement of subsurface radioactivity of those formations in the Plains which exhibit poor outcrop.

Well-log records available at the Research Council of Alberta were sampled for every sixth Township and fourth Range begining at Township One, Range One (McDonald and Van Dyke 1979). In total, 321 logs were assessed, providing a sampling density of one log per 795 square miles over the whole Province. For each gamma-ray log evaluated, the following data were collected:

1. The gamma-ray log name and location.
2. The drilling record, including the hole depth, diameter, mud density and depth of formation tops.
3. The casing record, including the interval cased and the cement thickness.
4. The logging record, including the logged interval, type of sonde and its configuration, and the units of measurement.
5. A measure of the background radiation over successive intervals over which it remains fairly constant.
6. A measure of any noteworthy high radiation peaks and the depth at which they occur.

This logging record provides a database of corrected gamma radiation levels of most sedimentary formations in the Province and is utilized by this study primarily for those formations which are potential source- or host-rocks of significant uranium mineralization. Because of this application of gamma-ray log data, a close and critical examination of the technique is warranted.

The use of these logs in uranium resource evaluations is complicated by the following difficulties:

1. Drilling- and logging-procedures affect the logged results.
2. Petroleum exploration targets often differ from favorable uranium exploration targets.
3. Gamma-logs do not provide a direct measure of uranium concentrations in the formation.

Those drilling and logging conditions which alter the measurable gamma radiation in the borehole are of two general types: Firstly, there are those factors which result from using different drilling and logging equipment, for which the logged results can be corrected to produce standardized data. These include corrections for borehole logging through well-casing, drilling muds of different densities, holes of different diameters and logging sondes

of varying design and sensitivity. Only in the occasional instance where incomplete drilling records were compiled, do these conditions present any difficulty. Where no complete logs were available, correction factors were arbitrarily assigned, based upon the practices of the logging company in that general area. If the corrected results were inconsistent with nearby borehole—logs over a large section of the logged interval, or where the results above and below the lower edge of casing vary for no apparent reason, the results were rejected.

The second class of drilling and logging error involves random changes of borehole conditions such as a changing hole diameter or the influx of formation fluid into the well. Of particular concern was the infiltration of radon gas at the end of the hole casing which produces an anomalously high radiation peak. These types of errors restrain any strict quantitative comparison of local peak gamma—ray log values from one borehole to another.

Logging records were obviously compiled only in those sections of the hole necessary for petroleum exploration. As a result, the logging coverage was barely adequate in some formations not favorable for petroleum exploration. Most significant in this regard are Tertiary and Upper Cretaceous sections in central and southern Alberta, including the Paskapoo Formation and the Edmonton Formation. In addition the logging record in notheastern Alberta, outside the domain of the Athabasca Tar Sands, is sporadic. For the

density of this study enough suitable logs were found, but any more intensive gamma-ray log studies will likely encounter this difficulty.

Scintillation counters are sensitive to a wide range of gamma-emitting radioactive sources including potassium-40, daughters in the uranium -235 and -238 decay series and the daughters of the thorium-232 decay series. Zeller et. al.(1976) examined the usefulness of petroleum industry gamma-ray logs in the identification of subsurface uranium provinces within the Morrison and Dakota Formations of the Central Great Plains of the United States. Their study was bounded on the east and west sides by the 99th and 104th meridians and on the north and south sides by the North Platte and Canadian Rivers respectively (covering the western half of Kansas and parts of Colorado, Nebraska, Oklahoma, Texas and New Mexico). Over three hundred wells were statistically evaluated and plotted using trend surface analysis. With regard to gamma-logs, Zeller et. al.(1976) concluded that they can definitely be used to identify potential uranium provinces with reservations and that:

> "Minor regional gamma-ray activity variations could be due to changes in the thorium content of the heavy mineral fractions or to changes in the potassium concentration which will vary with the clay content or arkosic nature of the formation. Any major increases in gamma activity are probably due to variations in the uranium content...."

This conclusion appears to contradict the interpretation of gamma -ray logs used by the petroleum industry, whereby they commonly assume that any major increase in the gamma-log

response is due an increased potassium-40 content, indicating the presence of pelite horizons.

Owing to this contradiction, a semi-quantitative evaluation of the gamma-ray spectrometric characteristics of McMurray Formation drill core was undertaken.* Lithological features such as the sandstone/siltstone ratio, the presence or absence of bitumen, carbonaceous material or pyrite and any alteration features were noted along with the total count (analagous with a scintillometer reading), potassium, uranium, and thorium spectrometer channel readings.** Over two thousand feet of core were examined from eighteen wellsites primarily from tar-sand bearing regions. It was observed that the total count spectrometer response generally increased with an increased shale content in the core, but that, owing to the unfavorable geometry this response was less sensitive than borehole gamma-ray logs. Additionally, the potassium, uranium, and thorium channel readings have no correlation with any particular lithological feature, although each of these channels was near its limit of sensitivity. It appears that the drill core was of insuffficent mass to produce a response similar to borehole logs, particularly for the potassium, uranium and thorium channels on such a small instrument.

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*Sections examined were from the ERCB core storage-facility in Calgary

**An URTEC UG-135 "minispec" spectrometer was used. This instrument contains a NaI(Tl) crystal of 4.0 cu. inches.

Owing to the general correlation between the total-count spectrometer results and the shale content of the formation and to the low contribution of the uranium and thorium channels, it can be concluded that in the great majority of cases in the McMurray Formation gamma-ray results do reflect the shale- or arkose- content of the formation. Where the gamma-ray log response of a given formation lies within the normal range of that rock type, any variation of the background uranium content will not be apparent. Only where the gamma-log response of a formation is anomalously greater than is typical for that rock type, can the results be attributed to an increased uranium content. For the purposes of this study any formation, or large part thereof, whose average gamma-response through the formation is anomalously larger than the response of a formation of similar lithology (particularly in shale and arkose content) or potassium concentration, has an inferred high-background uranium content.

Carrigy (1972) studied the lithology and chemical composition of the Paskapoo Formation and the Edmonton Group and found that both formations are relatively similar. * In particular, the average K_2O concentrations of the Paskapoo Formation and Edmonton Group were determined to be 1.83% and 2.19% respectively. The range of K_2O contents north of Red Deer (in the area where gamma-log data are available) were

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*In Carrigy's study as in this report, the lower boundary of the Paskapoo Formation is defined as the top of the Kneehills Tuff.

measured to be 1.26% to 2.87% in the Paskapoo Formation, and 1.77% to 2.55% for the Edmonton Group.

Based upon these similarities, this study would assume that any region of either formation that exhibited an anamalously high gamma-log response should qualify as a high background uranium area. Tabel 4. shows the average gamma-log response of the Paskapoo Formation and Edmonton Group from 10 wellsites between Township 43 and 61. It was calculated that the mean gamma-radiation level of the Paskapoo Formation and Edmonton Group are 69.1± 19.0 and 68.4±36.3 API units±one standard deviation. As no average gamma responses exceed the mean plus two standard deviations and as both formations have equal mean values, these units are assumed to have a normal radiation backgrounds and uranium contents for their rock types.

Table 4. Average radiation levels (API units) in selected drillholes of the Paskapoo Formation and the Edmonton Group.

Location	Edmonton Group	Paskapoo Formation
3-17-43-1W5	74	-
10-27-43-4W5	62	-
7-1-43-7W5	93	85
10-23-43-10W5	48	48
2-13-49-7W5	61	61
2-28-49-13W5	74	74
11-11-49-17W5	74	74
8-8-55-13W5	58	-
14-17-55-19W5	63	63
7-4-61-2W6	92	92

Other potential host-rocks in the Province are not as well documented with regard to their chemical composition. It is known that the McMurray Formation sands are composed

predominately of quartz (90% or greater, from Carrigy, 1973). It therefore could be predicted that the average gamma-response indicated on a drill log would be lower than that of either the Paskapoo Formation or the Edmonton Group. The calculated mean background gamma response in the McMurray Formation of 49.2 ± 18.2 API units$\pm$one standard deviation confirms the aforementioned prediction. Additionally no sample sites were anomalously high, which indicates that no uranium anomalies can be distinguished by this means in the McMurray Formation.

G. Groundwater Geology

Groundwater movement through strata is the primary mechanism for the transportation of uranium to a site of deposition and for the formation of epigenetic uranium deposits. Two general approaches for examining groundwater patterns were used in this study. By comparing groundwater patterns in Alberta with groundwater systems around known epigenetic uranium deposits, promising areas in the province might be identified. With the help of Research Council of Alberta hydrogeological reports it is possible to recognize:

1. Recharge and discharge regions.
2. The current movement of groundwaters around potential uranium source rocks.
3. Variations in groundwater chemistry which might mobilize or precipitate uranium.

The results of this pilot investigation are reported in the context of the aforementioned uranium evaluation "districts."

The second aspect of the pilot study was an attempt to determine typical uranium concentrations in some Alberta groundwaters. Three techniques of analysis were evaluated and are discussed below. These included fission–track analysis, uranyl–ion fluorescence analysis, and uranium adsorption by activated charcoal.

Fission–track analysis

Irradiating a sample solution with a controlled neutron flux can induce fission of ^{235}U atoms. The fission products produce detectable "fission tracks" in "Lexan" polycarbonate plastic immersed in the sample solution. The "lexan" plastic is subsequently etched in 6N sodium hydroxide and the tracks counted, using a microscope (see Appendix 3 for a complete experimental procedure). Given a constant neutron flux and irradiation period, the number of fission particles penetrating any given area of plastic is proportional to the concentration of ^{235}U in the solution and the total U content, in turn, is thus proportional to the deduced U–235 content.

An assessment of this technique was undertaken to determine the feasibility of measuring groundwater uranium concentrations using the SLOWPOKE reactor facility at the University of Alberta. It was hoped that a procedure for analyzing large sample populations might be achieved. Such a technique would be considered practical if:

1. The sample preparation technique was relatively clean and simple.
2. The analysis time was relatively short.

3. The number of fission tracks produced was great enough to provide a lower limit of detection of 0.5 to 1.0 ppb.

Seven test solutions were prepared, ranging in concentration from 386 ppm to 0.01 ppb. An eighth blank sample was included in all test runs. * The first test run in the reactor for a period of 1000 seconds, showed that the technique was inadequate for analyzing both highly concentrated and dilute uranium solutions (see Table 5 and Figure 12).

The spread in the standard deviation of the 19.65 ppm U solution indicates an error of ±7 ppm U. This error is largely due to the high density of fission tracks which causes counting errors. For a solution containing 1 ppm U, the indicated error is ±0.6 ppm U. The error for solutions with 50 and 2.5 ppb U is ±40 and ±3 ppb U respectively. An increase in accuracy would be possible if the 'unit area' for fission track counting was enlarged. This would appreciably increase the operating time though.

Sucessive reactor runs of 2000 and 3000 seconds showed a variable, but generally decreased sensitivity of the technique, due to irradiation damage of the Lexan plastic. From these data it is apparent that the technique lacks the necessary sensitivity under the maximum neutron flux allowed for the University of Alberta SLOWPOKE reactor.

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*The following solutions concentrations were prepared: 1. 386.5ppm 2. 19.65ppm 3. 994.6ppb 4. 50.44ppb 5. 2.557ppb 6. 0.13ppb 7. 0.01ppb

Table 5 Analysis of uranium in solution using the fission-track technique, with irradiation periods of 1000, 2000, and 3000 seconds and a neutron flux of 10^{12} neutrons/cm^2.

1000 secs.

Uranium conc. (ppb. U)	Av. no. of tracks per unit area (1mm X 0.5mm)	Total no. of fission tracks	Std. Dev.
19,560	332.40	1662	42.17
944	80.00	1120	35.93
944	83.14	1163	30.73
50	13.30	479	6.87
50	14.00	322	7.18
2.5	2.91	128	3.10

2000 secs.

2.5	0.8	45	1.46
2.5	0.2	5	–

3000 secs.

2.5	3.25	169	5.04
2.5	1.69	87	1.42
0.13	0.08	3	–
0.13	0.07	3	–
0.01	0.02	1	–
0.01	0.09	5	–

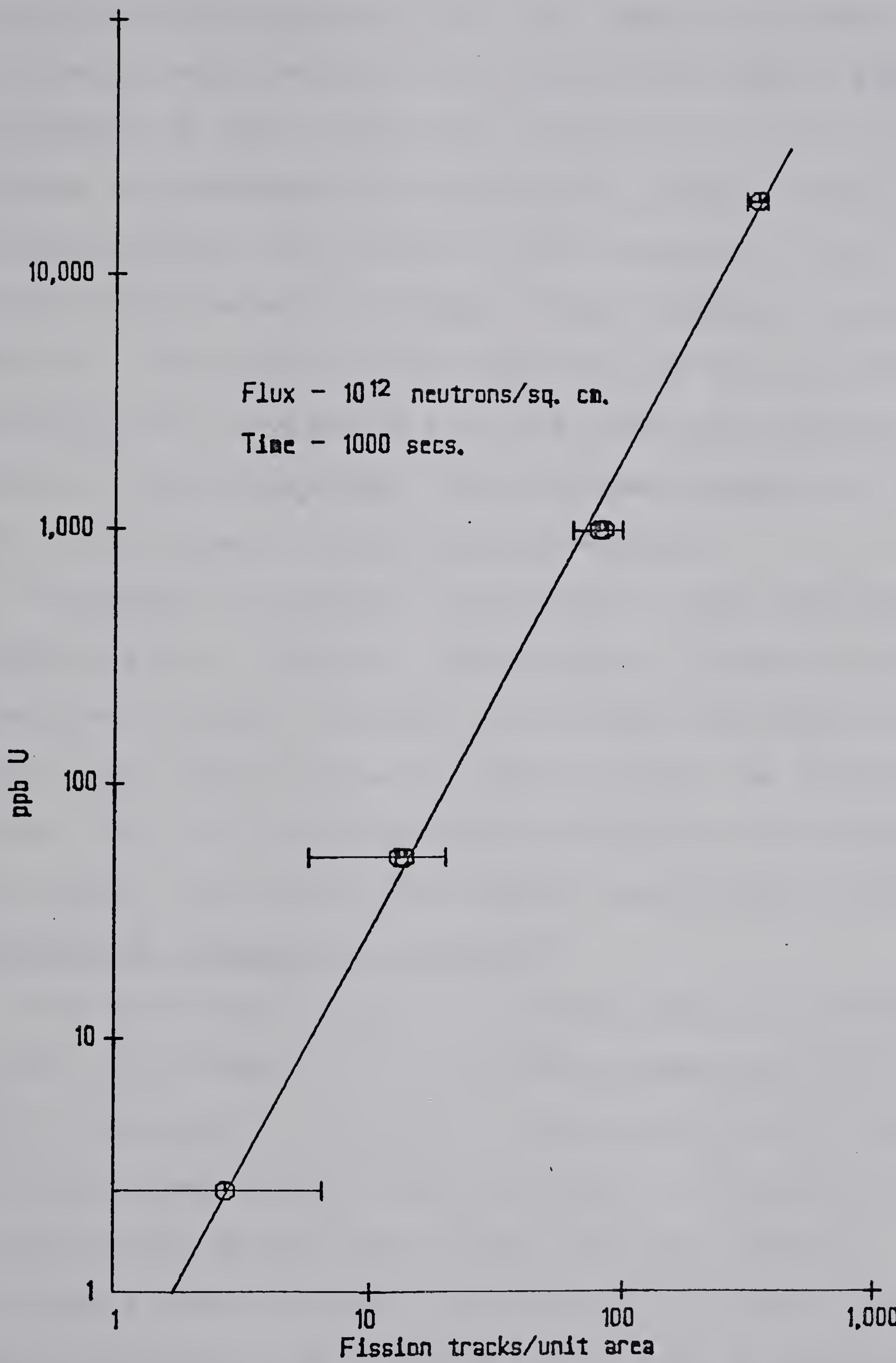

Fig. 12. Log–log plot of U ppb. vs. fission–track count per unit area.

With regard to criteria (1) and (2), concerning the simplicity of the technique and the speed of analysis, a few shortcomings were encountered. These would require some adjustments of the experimental procedure if fission track analysis were pursued on a large scale. Water leakage from the small plastic vials used was an annoyance in this limited study and would present a major problem in a larger operation. Melting the lip of the vial top onto the sides of the vial using a soldering iron was a delicate process achieving variable success. This problem proved to be the only tedious aspect in the whole procedure.

The major bottleneck with respect to time was that the SLOWPOKE reactor allows no greater than 10 samples to be irradiated at once. Following irradiation the sample is isolated for two to three days before it can be safely handled. This difficulty could be minimized and overcome with proper organization and remote handling facilities.

Ultra-violet fluorescence analysis

One of the most prominent uranium analysis techniques is that of fluorimetry. This method is based upon the fact that a uranyl-ion in solution or in a fused fluoride mixture produces a green visible light emission in response to stimulation by ultra-violet light. Scintrex Limited developed a laser-induced fluorimeter (the Scintrex UA-2 Uranium Analyser) with a stated lower limit of detection of 0.05 ppb U and measured accuracy of plus or minus 15% at 1 ppb U or above. The machine achieves this high sensitivity

primarily through the use of a high intensity ultra-violet source, namely a nitrogen laser. Quenching (light absorption) characteristics of elements such as iron and manganese are strongly decreased by adding a phosphate reagent buffer to each sample. The other major problem of organic fluorescence is compensated for by using a green optical filter and by electronically eliminating any short-term fluorescence (see Appendix 4 for a complete description of the analytical technique). *

A group of groundwater samples from the Milk River Formation were analyzed using this instrument, the results are summarized in Table 6.

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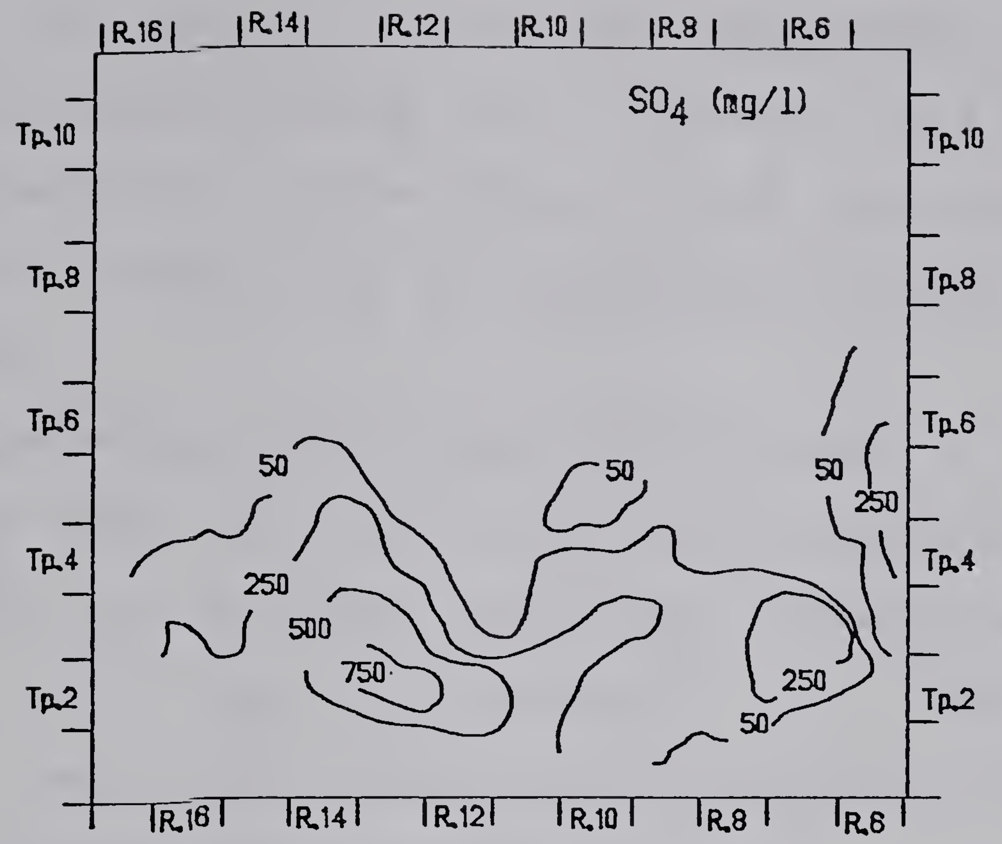
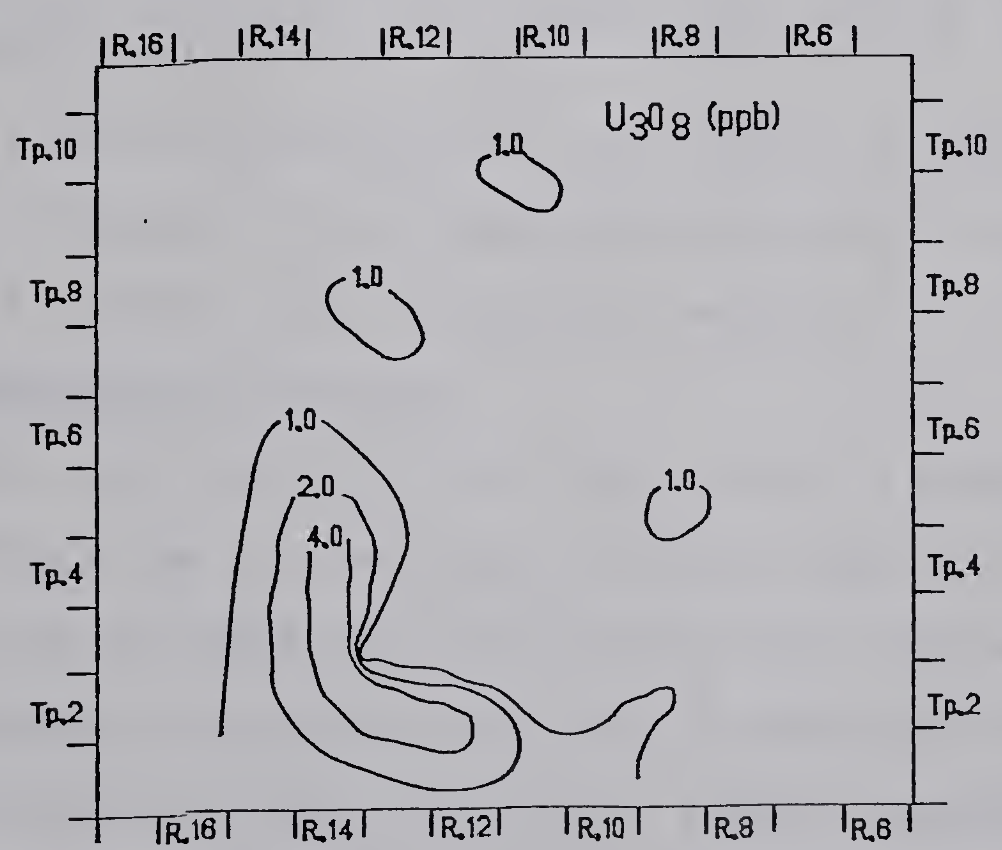
*Uranium is characterized by a relatively long period of fluorescence.

Table 6. Uranium content in the Milk River aquifer.

Sample	Location; (W 4th Mer.)	ppb U
MR 2	NW-21-4-11	und.
MR 3	SW-11-5-12	und.
MR 4	SE-6-5-13	1.88
MR 5	SE-32-4-15	0.86
MR 6	SE-10-7-16	0.23
MR 7	NW-12-5-10	0.56
MR 8	SW-18-6-10	und.
MR 9	NW-33-7-11	0.44
MR 10	SE-28-9-12	0.33
MR 11	SW-11-10-12	0.55
MR 101	NW-13-9-17	und.
MR 102	SE-10-9-15	0.41
MR 103	NE-28-9-13	0.53
MR 104	NE-35-9-11	1.06
MR 105	SE-26-9-9	und.
MR 201	NW-14-8-7	und.
MR 202	SW-20-7-13	und.
MR 203	SE-4-8-13	1.61
MR 205	NW-2-8-9	0.58
MR 301	SW-12-6-17	0.41
MR 302	SW-34-5-14	1.67
MR 303	SE-10-6-13	0.35
MR 304	SW-4-6-11	und.
MR 305	NW-12-5-9	1.10
MR 306	SW-2-6-7	und.
MR 401	SW-25-3-17	0.63
MR 402	SW-13-4-14	4.09
MR 403	SE-9-3-13	0.45
MR 404	SE-7-3-10	0.48
MR 405	NW-7-4-8	0.23
MR 406	SW-7-4-6	0.76
MR 501	SW-12-2-16	0.56
MR 502	NW-31-1-14	2.01
MR 503	NE-15-2-13	4.81
MR 504	NE-8-1-10	1.24
MR 505	SE-18-1-8	0.33
MR 506	NW-14-2-4	und.

und.=undetected





Figs. 13 and 14. Distribution of sulfate- and uranyl- ions in the Milk River aquifer (sulfate-ion distribution map from Schwartz and Muelenbachs, 1979).

Each sample was analyzed on two or three separate occasions. The range of values from the average was consistently plus or minus 7 to 15 percent. This is the indeterminate error of the analysis which falls within the specifications defined by the manufacturer of the fluorimeter.

The potential effectiveness of this tool is illustrated by figures 13 and 14, showing the areal concentration distribution of the uranyl and sulfate (from Schwartz et al., 1979) ions in solution. Schwartz et al. (1979) suggested two processes which could decrease the sulfate ion concentration as shown in figure 13:

1. Sulfate reduction and the formation of hydrogen sulfide gas.
2. Dispersion by mixing with other water with a low sulfide ion concentration.

Whichever process predominates, the results of the pilot analyses illustrate current uranium transport and reduction activity in the Milk River Formation aquifer.

Uranium adsorption technique

Van der Sloot et al. (1975) successfully analyzed low concentrations (as low as 0.4ug U/litre=4 ppb U) of uranium from solution by adsorbing the uranium onto charcoal and then subsequently analyzing the uranium-bearing charcoal by neutron activation. They developed a simple procedure whereby a high and constant percentage (95-98%) of uranium could be adsorbed by the charcoal. After reviewing the technique with Atomic Energy of Canada Ltd., it was decided

that its successful application depended upon:

1. Having a uranium-charged charcoal sample large enough to fall within the detection limits of AECL facilities.
2. Obtaining a charcoal with a very low background concentration of uranium.

Optimum adsorption conditions determined by Van der Sloot et al. (1975) utilized 200-500 ml. of water and 0.1 mg/ml. charcoal. For a 500 ml. sample containing 1 ppb uranium, the charcoal could recover 0.5 ppm uranium (with 100% uranium adsorption). Charcoal samples (Merc) sent to AECL had a background concentration of 0.12 and 0.13 ppm uranium. These results produce a significant error in the determination which may vary from one sample to another. In addition, a 0.5 gram charcoal sample would appear to be barely adequate if not too small as AECL stated that its limit of detection for a 1.0 gram sample was 0.08 ppm uranium.

Pre-treating the charcoal or using larger volumes of groundwater were judged to make the charcoal adsorption technique too cumbersome for use in such studies.

IV. Economic Study of Mining Sandstone Uranium Deposits

A. The PRICE2 program

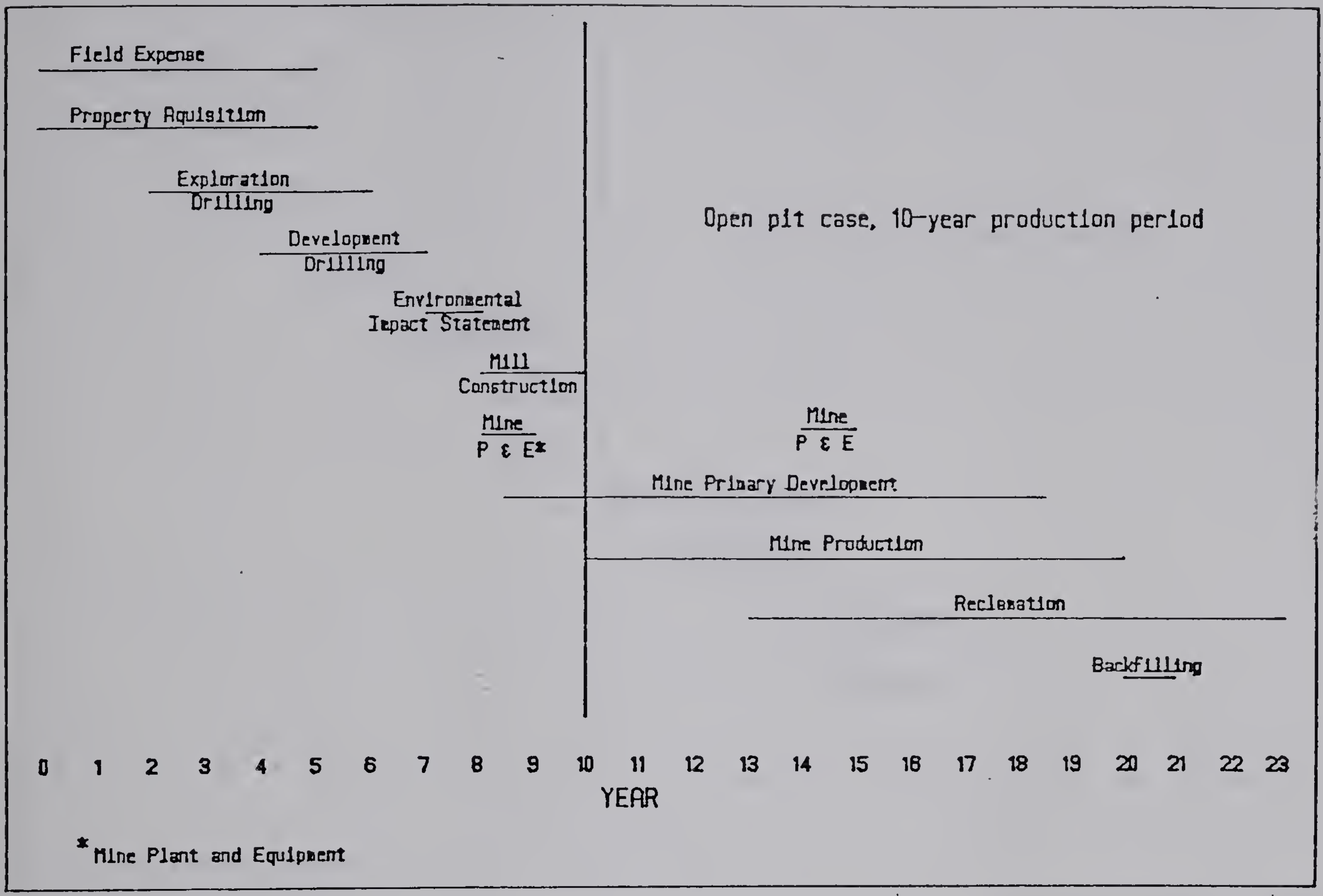
The PRICE2 program is an adaptation of the PRICE.FOR computer program developed by Trevor R. Ellis at the Colorado School of Mines; Ellis (1979). Its purpose is to calculate the required selling price of U_3O_8 (yellowcake) to yield a specified rate of return on investment from a tabular sandstone-hosted uranium deposit, characterized by a given depth to thickness ratio. The minimum required selling price of U_3O_8 is calculated using the "Net Present Value" (NPV) evaluation technique. Given a required rate of return, PRICE2 calculates the discounted selling price of uranium to equal the total discounted costs. Only open-pit and underground-mining techniques are considered in the computations, as insufficient published data exist to permit consideration of the economic viability of in-situ leaching methods at present. Users should become well acquainted with Ellis' paper before attempting to run the PRICE2 program. Appendix 5 includes the following:

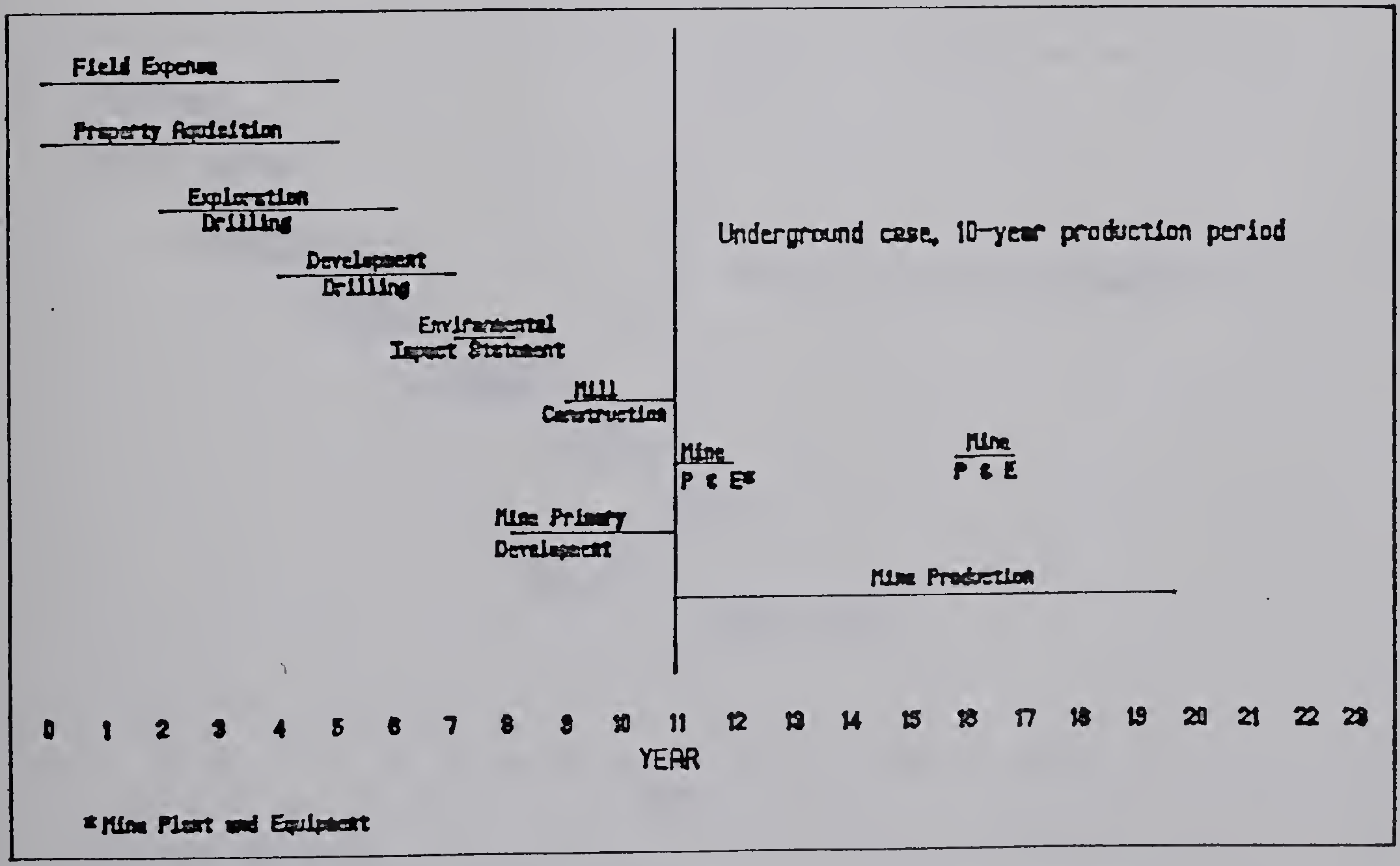
1. A copy of the PRICE2 program.
2. An expanded explanation of input parameters than what is included in Ellis' Users Guide.
3. An explanation of how to run the program.
4. A compilation of PRICE2 results utilized in this report.

Three general types of information are utilized in the PRICE2 program which are discussed in theory and in specific context to this study in Alberta, namely: time frames, physical information and cost considerations.

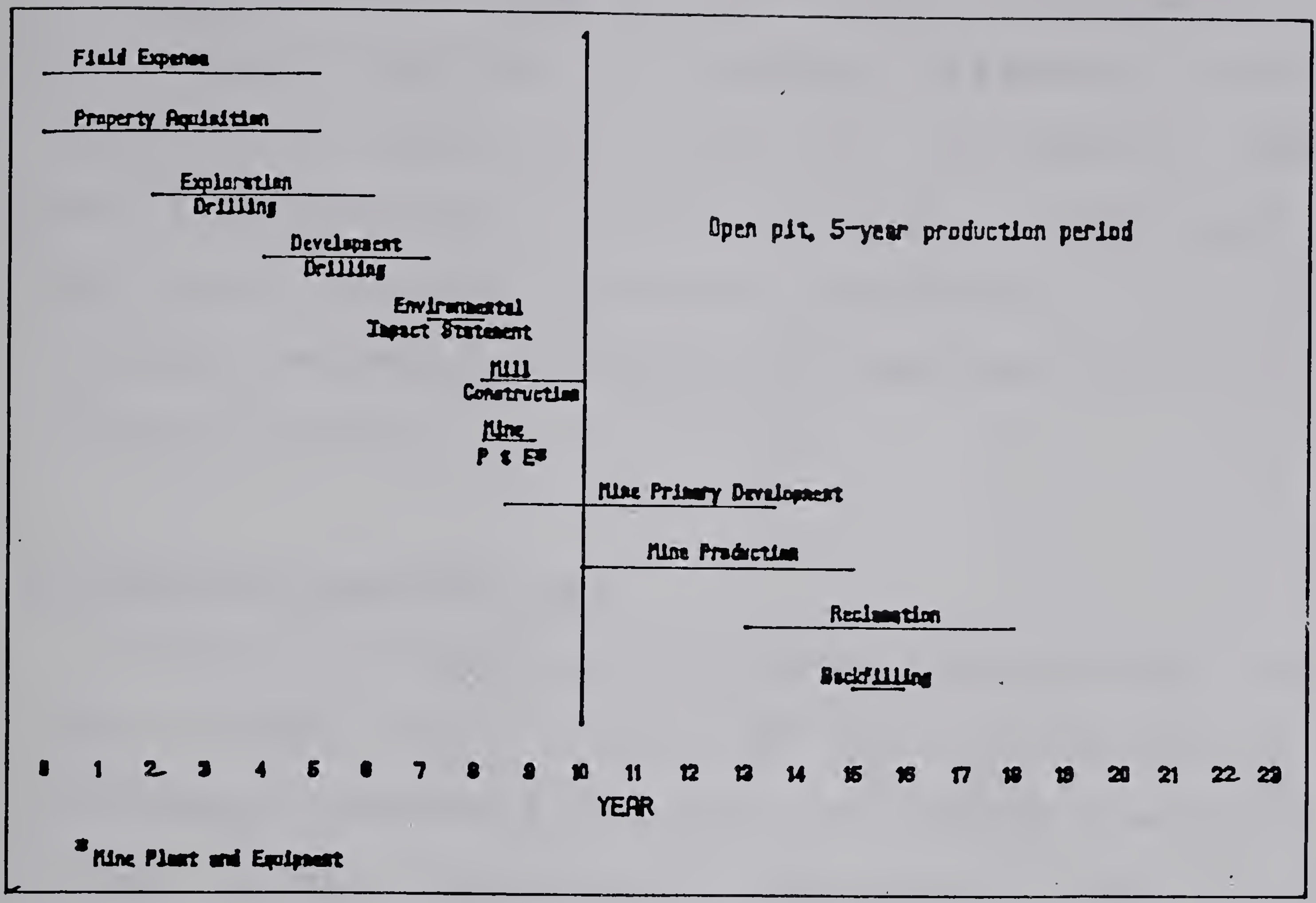
B. Time Frames

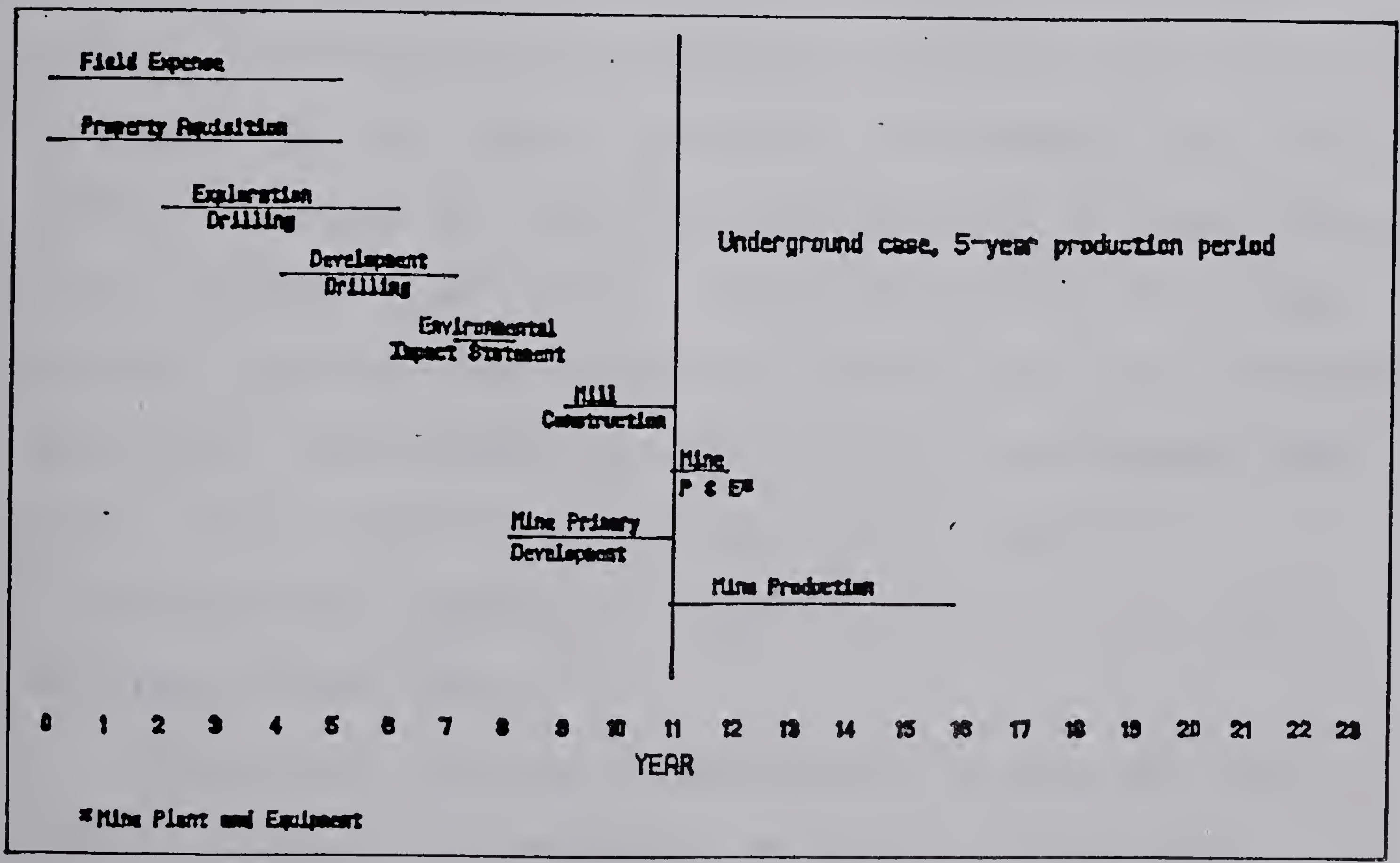
Most parameters in the PRICE2 program are used in the context of one of four time frames designed to simulate a deposit's life from the begining of exploration to the final stage of reclamation. A five and ten year mine production period model was developed for both undergound and open–pit operations (see figures 15 to 18). The importance of these time frames is difficult to overemphasize as they affect all discounting calculations. Discounting is a means of projecting current costs of exploration, development and operation to the period defined by the time frame of when these costs will be incurred (see 'Cost Information'). Therefore, variations from these time sequences of a mine's life will alter the calculated required price of uranium at a given rate of return on investment for a deposit. The time frames used were based upon Ellis's (1979) study, with the only major difference being that this study assumes that year zero is prior to any work being undertaken, while Ellis assumes year zero occures after some early exploration has been done and the economics of whether further work should be undertaken is being considered.





Figs. 15 and 16. Open pit and underground 10-year time frames (from Ellis, 1979).





Figs. 17 and 18. Open pit and underground 5-year time frames.

Mine operating lives of 5 or 10 years should apply to most deposits which might be developed in Alberta. As no uranium-mining camps presently exist in the Province, very small high-grade ore bodies are unlikely to be developed where no milling facilities exist. Only where an unusually large uranium deposit is found will these mine models be unlikely to apply.

C. Physical Considerations

Physical considerations are those which determine the deposit's size, depth of burial and thickness. These are the only general parameters describing the deposit. The PRICE2 program is able to accomodate a wide range of conditions as no limits are applied to physical parameters. Current economic and mining conditions on the other hand, pose some difficulties with regard to mining thin-bodied ores. Ellis (1979) suggests that open-pit mining costs increase rapidly with ore bodies less than 3 feet thick. For underground mining, operation expenses mount steadily as ore thickness decreases, and increase rapidly for ore thicknesses less than 6 feet. Data input for this study would be inappropriately applied to deposits thinner than the aforementioned limits.

Escalating operation costs due to increased depth of deposit burial are considered by Ellis (1979) to be insignificant for open-pit mines. They do increase for underground deposits thus Ellis applied a 12 percent

increase in mining costs with a doubling of the depth to thickness ratio. As this is the most up to date information available, it was adopted in these calculations.

PRICE2 calculates the deposit size using the "Mill Production Rate" (TPD.), the "Mine Life" (MLIFE in years.), and for underground mining UGDATA(8) a factor to compensate for ore left behind as pillars. For open pit and underground mining the tonnage of a deposit is calculated using equations (1) and (2) respectively:

1. $\text{TONNAGE} = \text{TPD} * 365. * \text{MLIFE} / 10^6$
2. $\text{TONNAGE} = \text{TPD} * 365. * \text{MLIFE} / (1. - \text{UGDATA}(8)) * 10^6$

UGDATA(8) is the percentage of ore left behind as pillars. Each of these parameters except UGDATA(8) has been varied to produce the combinations of deposits shown in Tables 7 and 8.

Table 8. Deposit tonnages evaluated for underground mining.

Mine Life (years)	Mill Rate (tpd)	Ore Grade ($\%U_3O_8$)	Deposit Size (tons*10^6)
5	1000	0.05	2.433
5	2000	0.05	4.867
5	3000	0.05	7.300
5	500	0.10	1.217
5	1000	0.10	2.433
5	2000	0.10	4.867
10	500	0.10	2.433
10	1000	0.10	4.867
10	2000	0.10	9.732

Table 7. Deposit tonnages evaluated for open-pit mining in Alberta.

Mine Life (years)	Mill Rate (tpd)	Ore Grade (%U 0)	Deposit Size (tons*10)
5	1000	0.05	1.825
5	2000	0.05	3.650
5	3000	0.05	5.475
5	500	0.10	0.912
5	1000	0.10	1.825
5	2000	0.10	3.650
10	500	0.10	1.825
10	1000	0.10	3.650
10	2000	0.10	7.300

It should also be noted that the computer program allows the user to specify the percentage of ore not recovered in underground mining operations. Ellis (1979) states that room and pillar mining is the primary method for underground mining of sandstone - uranium deposits, resulting in a loss of 20 to 30 percent of the ore reserves. This loss is accounted for in all pertinent cost and profit calculations.

D. Cost Considerations

Cost data are entered into PRICE2 in average dollars per short ton of ore produced. This format provides a consistent unit for comparing costs associated with deposits of varying physical characteristics. It also enables the user to utilize data from Klemenic (1974) which provide the most modern-large scale economic analysis of mining

sandstone—uranium deposits.*

The major calculation undertaken with these parameters is to discount costs using the Marshall and Swift Index, and the General Wholesale Index. The Marshall and Swift Index is applied to the expense of primary mine development, mill construction and equipment purchases. The General Wholesale Index is used to update field expenses, and the costs of property acquisition, exploration and development drilling. PRICE2 multiplies each cost variable by the ratio of the estimated index for the date at which the cost output applies (1990) to the value of the index for when cost inputs were taken (1979). The estimates of the future index values are derived by graphically extrapolating their growth patterns over the last ten years (see figures 19 and 20). This same process was utilized to update Klemenic's 1974 data to 1979.

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*The PRICE2 program also allows the option of entering exploration and development drilling costs on a dollar per foot basis.

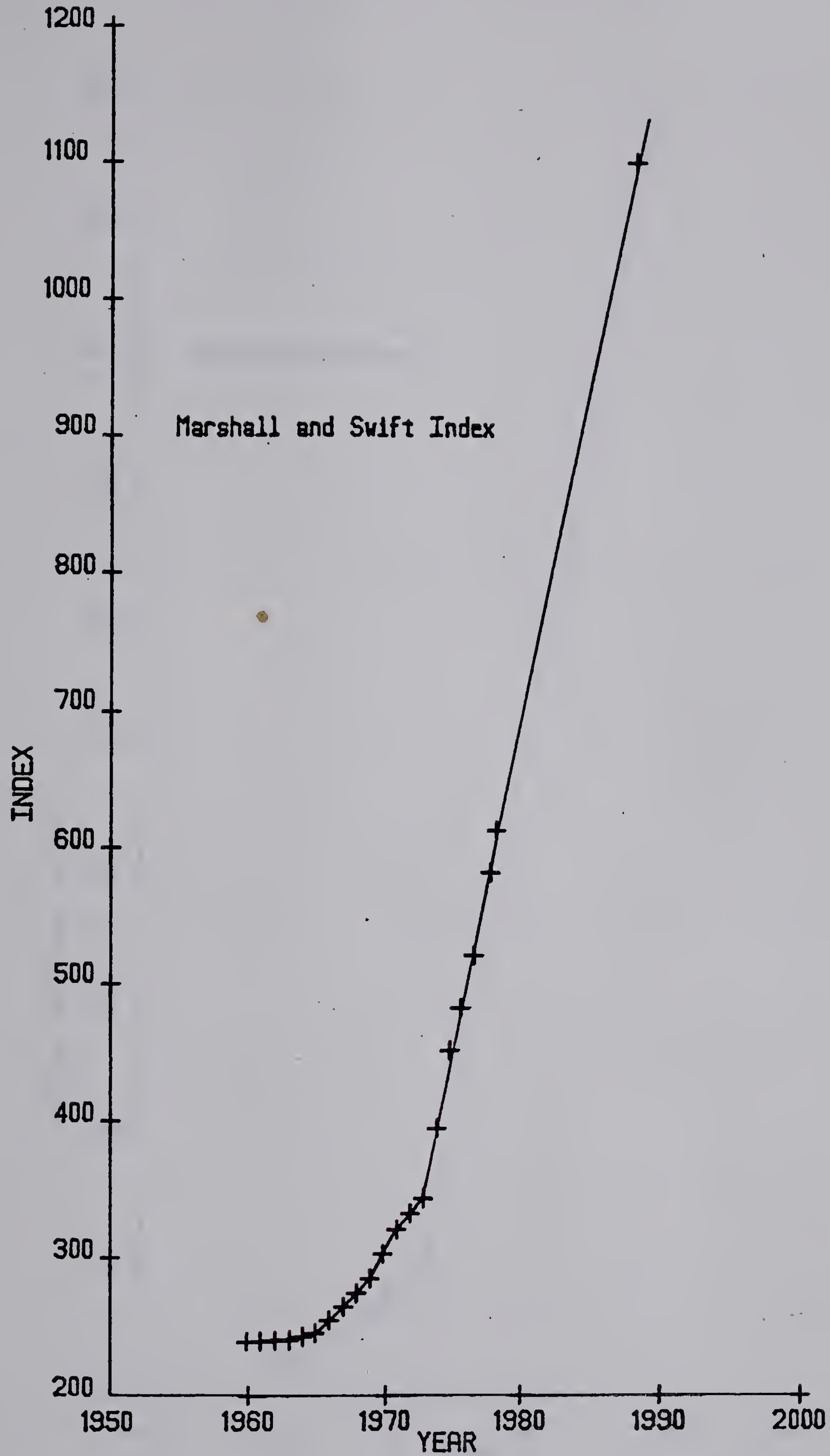

Fig. 19. Projection of the Marshall and Swift index to 1990.

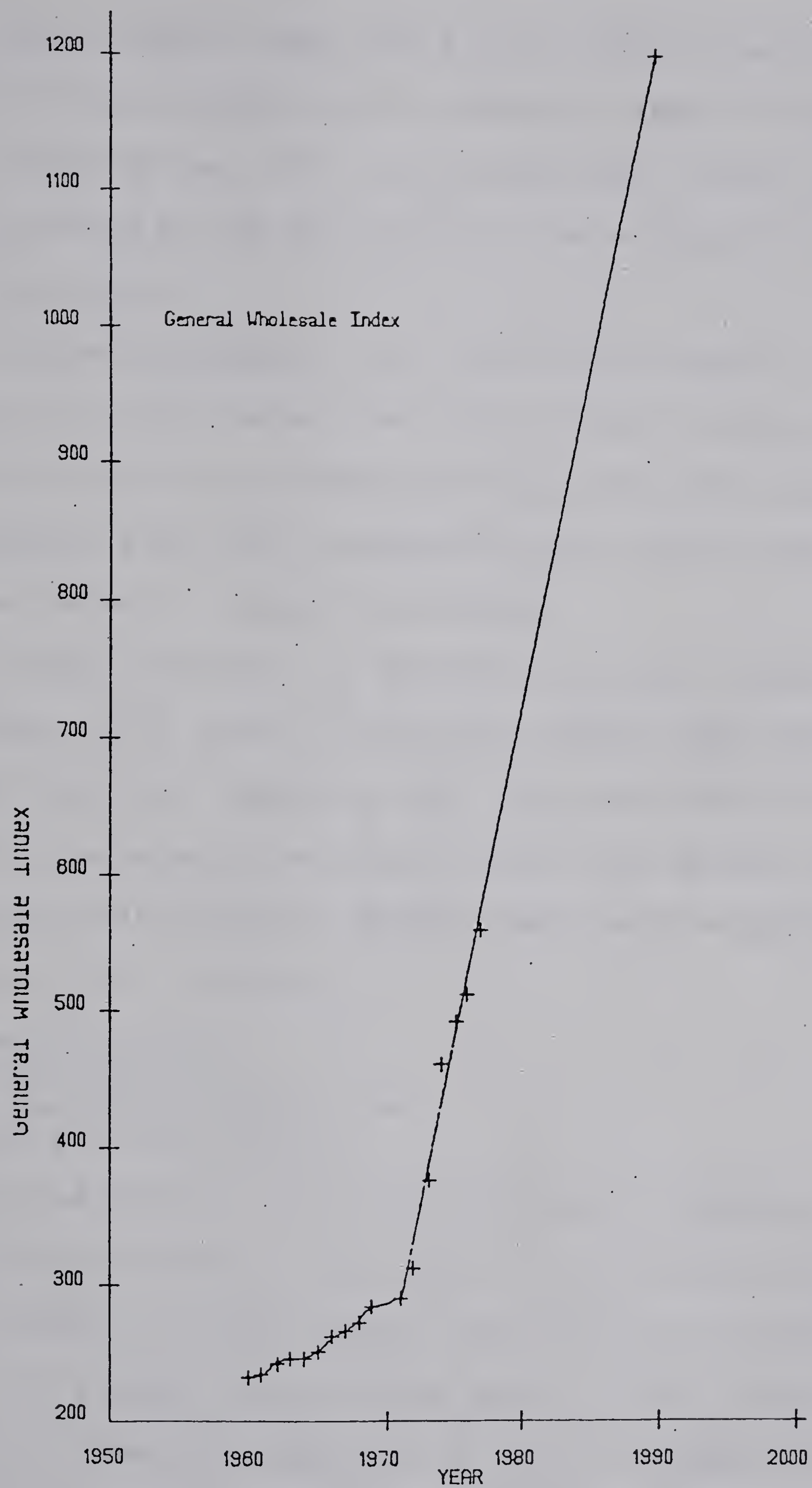

Fig. 20. Projection of the General Wholesale index to 1990.

Neither price index may be legitimately applied to the expense of performing an environmental impact study. For this study, Ellis's 1977 costs have been adopted and these are increased by 10% per annum to the appropriate year in the time frame.

Another assumption, not explicitly stated in the program, is that the mill site is within 5 miles of the mine and that the mill processes ore only from the deposit being considered. Both these assumptions are likely cases if a uranium deposit is found in Alberta.

One key parameter in PRICE2 is the ore "depth to thickness ratio (D/T)." This study adopts the same assumptions with regard to cost variations due to a changing D/T ratio as adopted by Ellis (1979). He assumes the following cost variables double when the ore depth to thickness ratio doubles:

1. Field Expenses.
2. Drilling costs.
3. Primary development costs.
4. Open-pit backfilling.

Underground mining operation costs were assumed to increase by 12 percent when the ore depth to thickness ratio doubles.

Klemenic's field expense and drilling cost data are valid for a depth to thickness ratio of 56.0. Other cost data from Klemenic, sensitive to the ore depth to thickness ratio, fall under either the open pit or underground mining models which are based upon ratios of 24.0 and 76.0 respectively.

Enviromental impact expenses can not validly be discounted using either index. Ellis for his 1977 model assumed that:

> "... the cost would be about $150,000 for a 500 ton per day open-pit mine and mill complex, increasing on a logarithmic scale to $500,000 for a 10,000 ton per day complex....that the cost for an equivalent underground case would be about two-thirds that for the open-pit case."

The only adjustment added to these assumptions in PRICE2 is that the expense increases 10% per annum from the year 1977 to the year the study is undertaken.

E. Uranium Pricing

Forecasting future uranium prices involves many complicated factors beyond the scope of this study. A brief review of the key factors and an estimate of the selling price of uranium at the date for which the PRICE2 output applies (1990) is useful.

Uranium is sold either at a "spot sales price" or a "contract price". The "spot sales price" is set by the Nuclear Exchange Corporation usually for immediate delivery of small orders of uranium. The price tends to fluctuate in reponse to the short-term demand for uranium. In the late 1970's this price was as high as $42 U.S./lb. U_3O_8, but it is currently selling at about $25 U.S./lb U_3O_8. The "contract price" is far more significant because it is the price upon which the economic success of the deposit depends. Data published by Morse and Curtin (1977) show the range and average sales price of uranium deliveries from

U.S. domestic production between 1976 and 1985 (see figure 21).

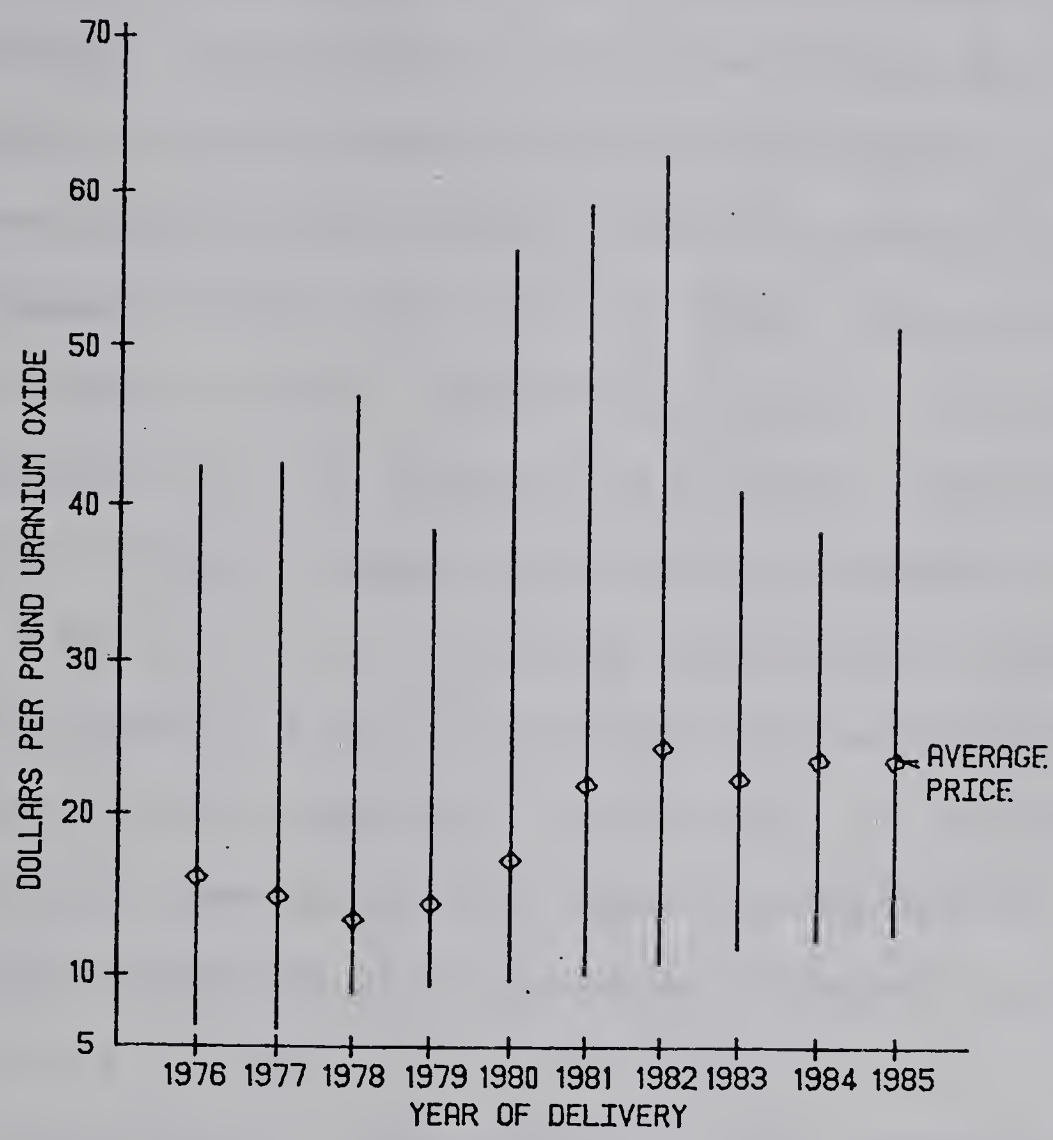

Fig. 21. Range of U.S. U_3O_8 contract sales prices (from Morse and Curtin, 1977).

Prices are quoted in year–of–delivery dollars, as estimated by the purchaser. The wide range exhibited for any given year is due to the fact that the price includes both fairly old and very recent contracts producing relatively low and high contract prices respectively. Nonetheless, the average contract price is roughly half that of the "spot sales price." Information on the "contract sales price" from recent contracts is unavailable but probably is in the

$30–35 range.

The boom in the price of uranium in the 1980's is directly attributable to the rapidly increasing price of petroleum. This trend in rising petroleum prices will probably continue because petroleum reserves are likely to decrease unless the current rate of consumption marketdly decreases (Gander and Belaire, 1978). The demand for growth in alternate energy industries is most likely to occur. In response to rising energy costs and the need for alternate energy sources uranium exploration expanded in the 1970's. As a result of this increased exploration a number of uranium deposits were discovered and developed in the last ten years. New deposits, particularly in W. Canada and Australia, have produced a temporary potential oversupply of uranium resulting in a decreased price of uranium in respect to prices in the mid– and late–1970's.

Additionally, the growth in the uranium power production industry has been slowed by many non–economic factors. In particular, the public's lack of confidence in the safety of nuclear power plants and concern over the environmental impact of mining and waste management resulted in an uncertain future in the nuclear power industry. Capital costs are high in all aspects of the uranium industry, particularly mining and power plant development, and these uncertainties make the economic risks very high. Recent studies by Neff and Jacoby (1981) predict hard times for the uranium mining industry, with production exceeding

demand. A seller's market is very likely to develop as producers accumalate large uranium inventories to at least 1990. Dahlstrom (pers. comm.), is less pessimistic in terms of the size of the production inventories and time period in which a seller's market will persist. Marginal producers will likely suffer the most in theses circumstances.

For this study, it is assumed that the nuclear industry will exhibit a slower economic expansion than the general economy over the next 10 years. An arbritrary three percent annual increase in the contract selling price is adopted by this study until 1990 resulting in a "contract price" of about \$40U.S./lb U_3O_8. After this period, the "contract selling" price is assumed to increase 5% per annum for a further ten years. During this 20-year period costs are assumed to increase 5% per annum.

F. Results and Interpretation

The computer analysis has been used to illustrate to what degree varying the conditions of mining a sandstone-hosted, uranium deposit affects the economic viability of the operation. In particular, it is possible to indicate:

1. What are the minimum conditions in terms of ore grade and tonnage which determine whether a deposit be economically mined with a required selling price of \$40 U.S./lb. U_3O_8?
2. At what ore depth to thickness ratios does underground minimg become more economically attractive than open-pit mining?
3. How do variations of the required selling price affect the economic viability of mining a deposit?

4. How does varying the mine life by varying the rate of production affect the economic viability of a deposit?

Exact conclusions are not possible with this type of investigation, as the conditions describing any given deposit are unique. These observations should only be considered as generalizations.

Figures 22 to 24 are graphs of the \$40 U.S./lb. U_3O_8 and the \$50 U.S./lb. U_3O_8 required uranium selling price for deposits of varying reserves versus depth to thickness ratios. Deposits with an average ore grade of 0.05% U_3O_8 are evaluated assuming a mine life of 5 years, while 5- and 10-year production periods are considered for deposits averaging 0.10% U_3O_8. The discounted required rate of return (DCFROR) in all these cases is 21%.

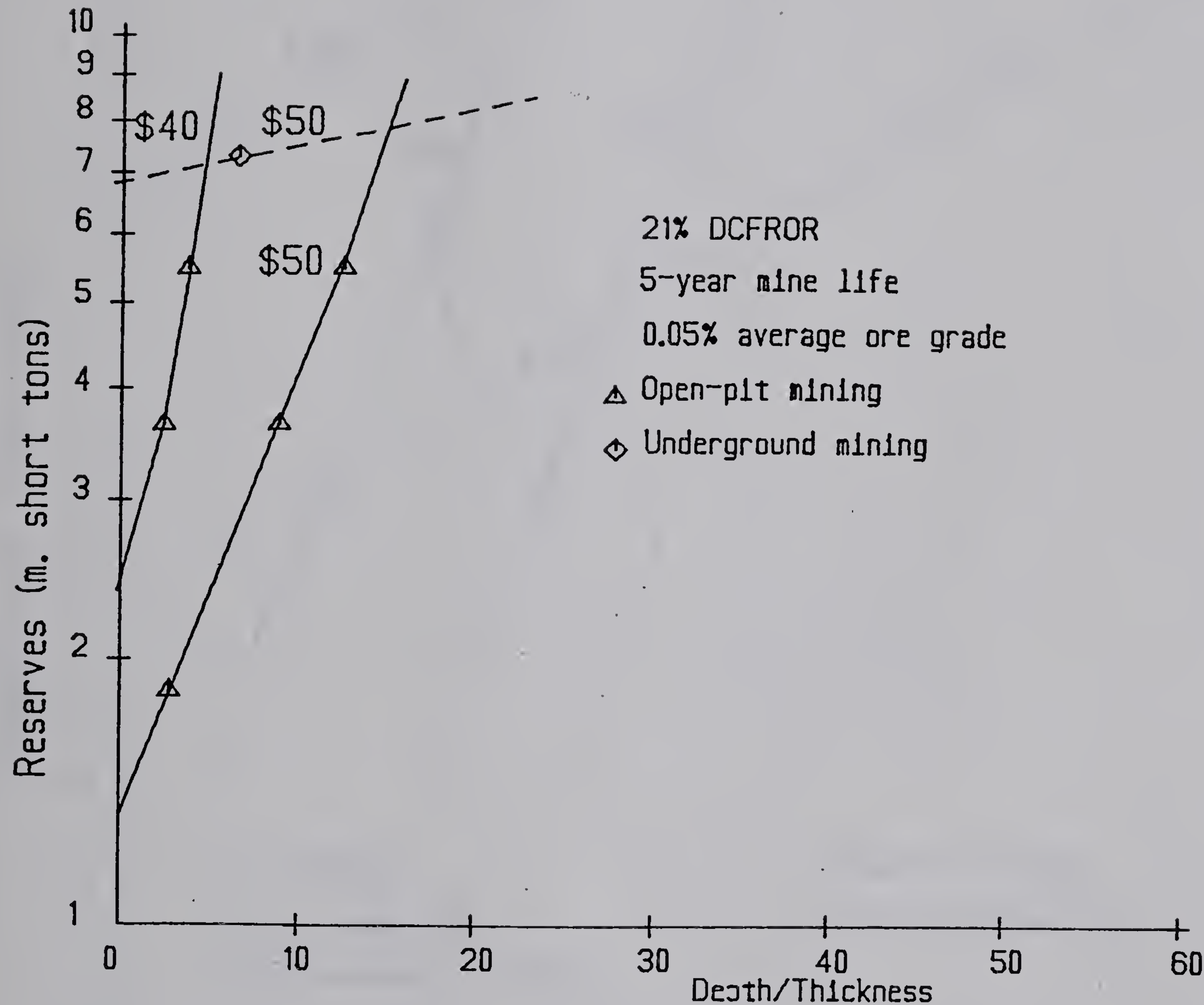

Fig. 22. Ore reserves vs. depth/thickness ratios of deposits with an 0.05% U_3O_8 average ore grade and 5-year mine life.

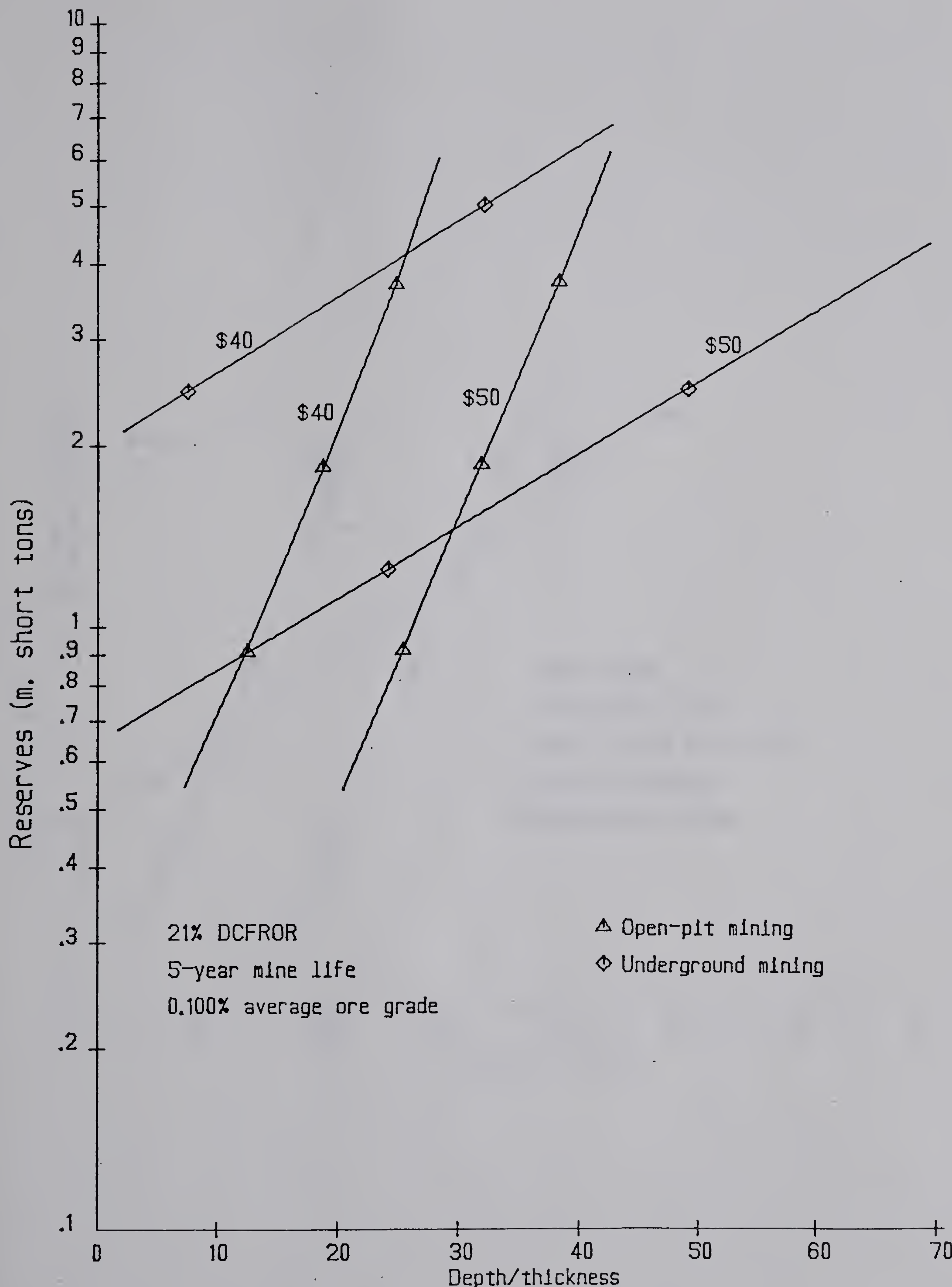

Fig. 23. Ore reserves vs. depth/thickness ratios of deposits with an 0.10% U_3O_8 average ore grade and 5-year mine life.

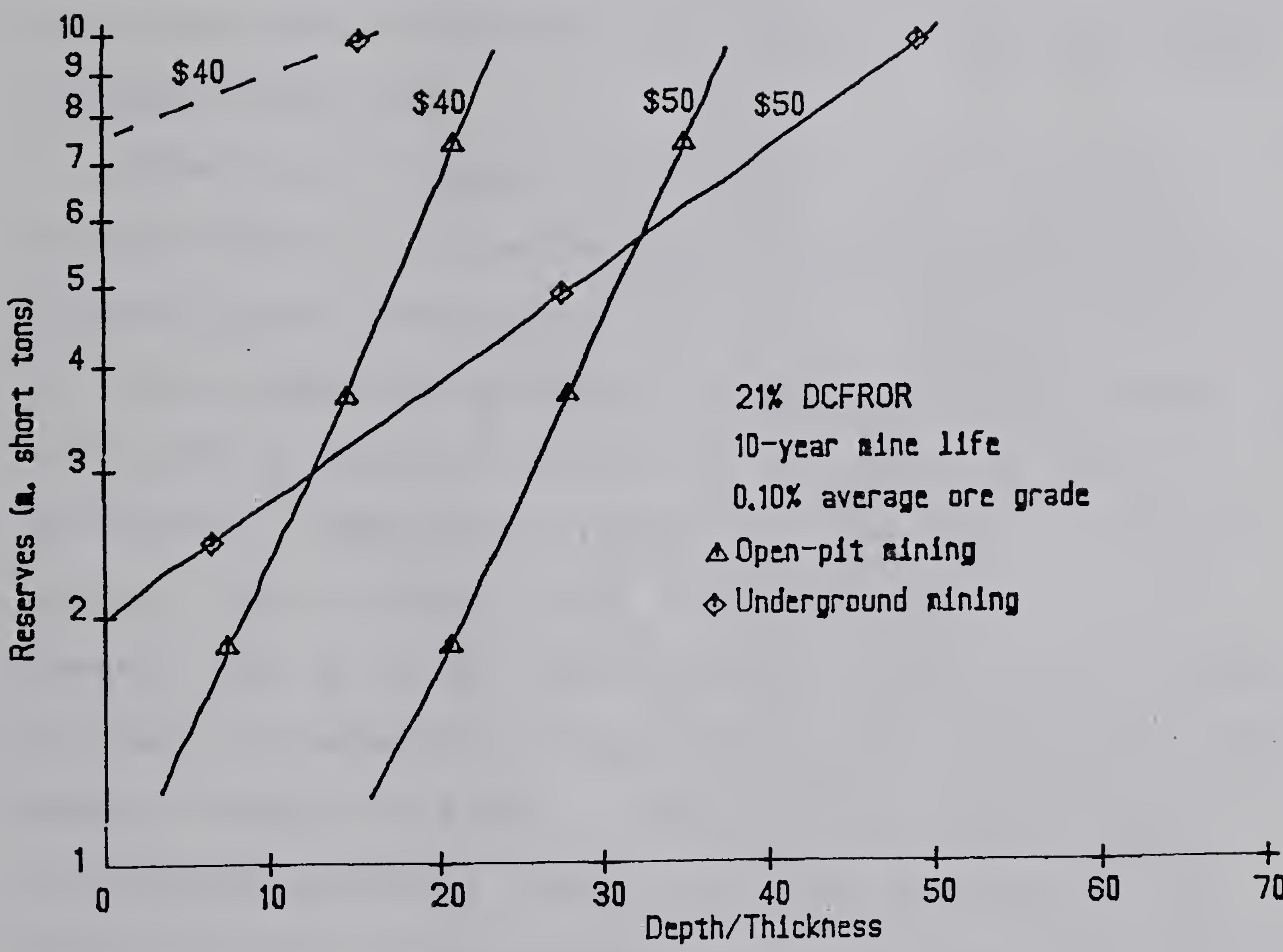

Fig. 24. Ore reserves vs. depth/thickness ratios of deposits with an 0.10% U_3O_8 average ore grade and 10-year mine life.

Figure 22 indicates that underground mining is likely to be uneconomical at a selling price of $40 U.S./lb. U_3O_8 for deposits averaging 0.05% U_3O_8 in the range of tonnages analyzed. For a deposit of this average ore grade an open-pit mining operation would be economically feasible if the ore depth to thickness ratio is less than 7. Most sandstone-hosted deposits of this grade would have to lie within 100 feet of the surface to be mineable under these economic conditions.

When deposits averaging 0.10% U_3O_8 are considered, open-pit mining is economically attractive with low- and medium-tonnage deposits (as small as 0.5 million short tons) if the ore depth to thickness ratio is 2 or 3 to 20 and the mine life is 5-years. Underground-mining would only be preferred to open-pit mining if the total ore tonnage is greater than 4 million short tons, and the D/T ratio is greater than 20 or 25. Given a deposit with a thickness of 20 feet, and moderate tonnage, an open-pit operation up to a depth of about 400 feet is possible. Below this depth underground mining is likely to be more economical. For deposits with a 10-year production period open-pit mining is the only economical means of mining the ore if the ore reserves is less than 10 million short tons. This is only economically attractive if the depth to thickness ratio is less than 20 to 25.

In general open-pit mining is more economically sensitive to the ore depth to thickness ratio, but where an

ore body has a low ore depth to thickness ratio, it is more likely to be economically mineable for low tonnages and grades. Underground deposits must have a higher minimum ore grade and tonnage before they are economical but they are less sensitive to variations of the D/T ratio. For these reasons any increase in the selling price of uranium will make open-pit mining more attractive at greater depths, and underground mining at lower tonnages and grades. Comparing figures 23 and 24 indicates that it is more profitable to maximize the scale of the mining and milling facilities at a higher initial cost and to recover the ore at a faster rate. It would be absurd to extrapolate this trend too the extreme, as this trend will undoubtably change. Additionally, the rate at which the yellowcake can be sold will affect rate at which the deposit can be economically mined.

V. Uranium evaluation districts in the United States

Two mining districts in the United States have been adopted for comparative purposes to permit evaluation of the uranium potentials of Alberta (excluding the Precambrian Shield). These are the Powder River Basin and the South Texas Coastal Plain. Each area exhibits features which to some degree are found in Alberta's strata.

A. The Powder River Basin

The Powder River Basin contains a Tertiary section disposed in a north–south trending asymmetrical syncline, with the deepest sections adjacent to western uplifts. The basin measures 12,000 square miles in area, but the most significant mineralization covers an area of 400 square miles in four mining districts. These are the Pumpkin Buttes, Turncrest, Monument Hill and Highland–Box Creek areas. Pre–1976 uranium production from the region was approximately 3000 tons U_3O_8 concentrate, most of which was mined in the early 1970's (Curry, 1976 and U.S. Bureau of Mines Minerals Yearbook, 1970–1975 inclusively). Reserves and total resources measured for the basin as of 1976 are 107,200 and 78,000 short tons U_3O_8 of \$30/lb concentrate (Curry, 1976).

Ore deposits in the Powder River Basin are found in the Eocene Wasatch Formation and the Paleocene Fort Union Formation. Both units are fluviatile arkosic sandstones with interbedded conglomerates and mudstones plus intermittent

lignite coal and coaly shale intercalations. In a general west to east direction, sediments progressively grade into finer lithofacies, which results in a corresponding decrease in permeability.

Uranium mineralization is dominantly found in paleo-point and channel bars in the transition zone between upgradient red hematitic 'altered' sandstones and down gradient grey-buff 'unaltered' sandstones. Mineralization in these deposits is predominatly uraninite, coffinite and pyrite, with a noted absence of V, Mo, and Se.

These deposits represent the 'Wyoming roll-type' ore bodies which formed epigenetically by the precipitation of uranium from groundwater flowing within the permeable aquifers.* The downward migrating groundwaters moved from a southwestward recharge area where sulfate-rich groundwaters leached uranium, either from overlying volcanic tuffs or from the arkosic material in the formations themselves. At the solution fronts it has been hypothesized that pyrite previously produced from sulfate-reducing bacteria reduces the sulfate-rich groundwaters causing the precipitation of uranium (Dahl et al., 1976).

It appears that the controlling factors in the formation of these types of deposits are:

1. The paleofacies of the fluvial deposition system.
2. Organic material distributed adjacent to major aquifers.
3. The overlying unconformity which allows uranium-bearing

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*The coefficient of permeability within the Wasatch Formation in the Powder River Basin ranges from 3 to 72 gal/day/ft^2 with an average of 30.

groundwaters to migrate into the fluvial sandstones.

B. The Texas South Coastal Plain

In this study, the Texas South Coastal Plain encompasses the subcrop area of the Eocene Jackson Group in the Karnes–Live Oak–Duval County region. This covers an area of approximimently 7,000 square miles. Past production has totalled 7,000 tons U_3O_8 concentrate (or approximently 5.4 million tons of ore*). Reserves estimated for 1976 amount to 43,900 tons U_3O_8. The United States National Resource Evaluation, Preliminary Report (USERDA, 1976) estimated a 93,000 ton "probable" resource and a 38,000 ton "possible" resource (all resource figures from Dickson et al., 1977).

Uranium deposits are found in the Eocene Whitsett Formation of the Jackson Group, the Miocene Catahoula Tuff and Miocene Oakville Sandstone. The deposits in all these units are believed to have derived their uranium from the Catahoula Tuff. In Karnes County, uranium deposits developed as a result of the downward migration of uranium in groundwaters into fluvial channel sandstone of the Flashing Clay and Dubose Members and into strand–plain barrier bars of the Derveesville and Tordilla Members. Each of these units is a more permeable sandstone enclosed in lagoonal and palludal mudstones and siltstones. Uranium reductants active in this formation include authigenic plant material and

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*This figure assumes an overall average ore grade of 0.15% U_3O_8, and mine and mill recovery rates of 90% and 95% respectively.

allogenic H_2S which migrated upwards along faults into the fluvial sandstone causing pyrite deposition. Uranium precipitation resulted when uraniferous groundwaters later reacted with the pyrite. Fluvial channel deposits in the Catahoula Tuff and overlying Oakville Formation contain the uranium deposits in both Live Oak and Val Counties. Neither formation contains any authigenic reductants and reaction with the pyrite is considered the sole means of precipitating uranium.

VI. Uranium Evaluation districts in Alberta

For this study the Province has been divided into seven evaluation districts determined by regional drainage patterns,in addition to the Foothills and Rocky Mountains. In each area, continental and nearshore-marine formations are identified and their potential for hosting uranium mineralization are discussed. For those formations lying within more than one evaluation district, a description of the geology of that formation is given only within the text describing the evaluation district in which the formation is of greatest importance. Within each evaluation district where the formation is repeated, details specific to that region are noted and a reference to the evaluation district under which the detailed description is provided. Where sufficient data have been gathered, the resource has been quantified. No attempt is made to quantify less understood formations, as the uncertainty is too great. Additionally, formations, or parts thereof, not occuring within about 1000 feet of the surface, are not discussed in terms of their potential uranium resources. Such formations are unlikely to undergo any uranium exploration in the foreseeable future, for exploration costs are very high and the risk is too great for an area with no previous uranium exploration.

It is beyond the abilities of this study to statistically calculate the error in each speculative resource estimate. The United States National Uranium Resource Evaluation (1976) provides no indication of the

error in their speculative resource estimates, and generalizes a plus or minus 20 percent error for the $10-15 reserve estimates. The speculative resource estimates in this report are only rough approximations in which a plus or minus 50% error would be quite acceptable.

A. The Oldman-Milk River drainage basin

This evaluation district is the southermost in the Province and it stretches in the Plains area from the international border as far north as Township 17. Included in this region is the Alberta section of the Cypress Hills. Table 9 summarizes the quantified uranium resources of the region.

Table 9. Summation of estimated uranium resources in the Oldman-Milk River drainage basin.

Formations/ Groups	Min. Ratio	Area U.S. (sq. mi.)	Area Fm. (sq. mi.)	U.S. Res. (s. tons)	Est. Res. (s. tons)
Milk River	19/57	7,000	360	181,900	3,000
St. Mary and Blood Reserve	27/46	12,000	1,500	188,200	14,000
Willow Creek	19/46	12,000	1,470	188,200	10,000
Paskapoo	23/46	12,000	110	188,200	900
Total					27,900

The Milk River Formation

This Upper Cretaceous formation in southeastern Alberta is composed of argillaceous and carbonaceous, grey-buff, fine-grained sandstone, interbedded with green and grey

shale, mudstone and lignite (Furnival, 1946; Myhr and Meijer-Drees, 1976). Meyboom (1960) indicates that the clastic sediment source was the Selkirk Mountains to the west and that a significant amount of clay-sized material is derived from weathered volcanic rocks. A type section cannot be defined in any one locality, as the formation exhibits a considerable lateral facies variation. In a north to south direction the formation changes from a deltaic coastal-plain facies into offshore marine mudstones and shales (Myhr and Meijer-Drees, 1976). Additionally, a NW-SE trend of clean sand lenses was formed either by sediment deposition by streams or ocean currents parallel to the shoreline.

Groundwater studies done by Meyboom (1960) and by Schwartz et al. (1978) indicate that the major area of groundwater recharge is in the Sweet Grass Hills of Montana. The subsequent direction of groundwater flow is strongly influenced by the regional structural geology and by the sedimentary facies pattern. The key structural features are the Sweet Grass Hills and Kevin Sunburst Dome in northern Montana and the Bow Island Arch in southern Alberta. These features cause the Milk River Formation to plunge northwards in a fan-like configuration, which induces a regional south to north groundwater flow direction. The aforementioned NW-SE sedimentological trend causes groundwater movement to shift in a northwesterly direction. These patterns are reflected in the sulfate- and uranium- ion distribution maps, figures 8 and 9.

The sedimentological pattern of the Milk River Formation would appear to be relatively similar to that of the Whittsett Formation in the Texas South Coastal Plains. It would therefore be conceivable that uranium mineralization could be present in the fluvial and barrier bar facies of the Milk River Formation. Based upon the distribution patterns of the sulfate- and uranium- ion species, in addition to the distribution of the fluvial and littoral facies units, the most favorable area to find uranium would appear to be within the area within Townships 1 to 6 and Ranges 8 to 17. This covers an area of 360 square miles. The mineralization characteristics of the Milk River Formation are tabulated in table 10.

The Belly River Group

The Cretaceous Belly River Group in the southern Alberta plains consists of the Oldman and Foremost Formations which form a wedge of continental clastic material thinning eastwards and passing into marine strata in Saskatchewan. Both formations are composed of alternating sandstones and shales that have a very limited vertical and lateral continuity. These formations are composed of sandstones with quartz, chert, and feldspar, with coarser units tending to be more feldspathic whilst finer-grained units generally contain more quartz. Carbonaceous material is variably scattered in the wedge including some large subbituminous and lignite coal seams, most notably the Lethbridge Member. Thin, discontinuous bentonites and

Table 10. Mineralization characteristics of the Milk River Fm. and the South Texas Coastal Plain mining district.

	Milk River Fm..		Texas	
	Description	Value	Description	Value
Deposit/ Occurrence	None	0	Mineralization (M)	10
			Outcrop rad. (O)	5
	U geochem. (G) anomaly	2	U geochem. (G) anomaly	3
			Trace metals (T)	2
Host Lithology	Clastic Rocks (QF)	3	Clastic Rocks (CQVF)	5
			Low-rank (L) coal	3
			Volcanics (FT)	3
Uranium Provenance	Volcanic (V) Granitic (G)	1	Volcanic (V)	2
Depositional Environment	Continental (CDL)	3	Continental (CD)	5
	Marine (L)	2	Marine (L)	2
Structure	Facies (B) boundary	2	Facies (B) boundary	2
	Fold (Z)	1	Fold (Z)	1
			Unconformity (U)	2
			Fracture (F)	1
Reductants	Plant debris (C)	2	Plant debris (C)	2
			Sulfide min. (S)	2
			H_2S (H)	2
Permeability	Good (G)	3	Good (G)	3
Alteration	unknown	0	Bleaching (B) Limonite (L)	2

Total	19	57

bentonitic shales occur in the group, but they do not seem to be significant enough to be considered as a major potential source-rock of uranium. General descriptions of both formations indicate that "unaltered facies" rock units predominate.

Based upon the limited data available, it appears that the Belly River Group should be considered a moderately favorable host sequence, most likely comparable with the Powder River basin district. Negative factors are that there is no evidence of an "altered facies" and no known uranium occurrences.

The St. Mary River and Blood Reserve Formations

The Upper Cretaceous St. Mary River and Blood Reserve Formations are grey and green coloured sandstones with interbedded siltstones, mudstones and thin coal zones (Carrigy, 1972, McGrossen and Glaister, 1966). Feldspathic sandstones of the Blood Reserve Formation were deposited in a shoreline environment which graded upwards into alluvial plain deposits of the St. Mary River Formation. Lithologically and stratigraphically these units are equivalent to the Edmonton Group, and they exhibit many common potential uranium-hosting characteristics. Most importantly, they share a common potential source of uranium, namely the Kneehills Tuff and adjacent bentonitic zones. Therefore the St. Mary River and Blood Reserve

Formations will have nearly the same mineralization potential as the Edmonton Group, whose only additional favorable factor is the fact that it has produced an airborne radiometric anomaly (see table 11).

The Willow Creek Formation

Like the St. Mary River Formation this clastic wedge of material is the southern Albertan equivalent of the Paskapoo Formation. As the Kneehills Tuff directly underlies the Willow Creek and Paskapoo Formations, the tuff zone is a highly favorable source of uranium in the lower parts of these formations. For the middle and upper parts of these units the importance of detrital volcanic material as a source of uranium increases. The relative abundance of material in terms of the type of source rocks they were derived from are shown below:

Paskapoo Fm.: metamorphic>volcanic>sedimentary.
Willow Creek Fm.: sedimentary>volcanic>metamorphic.

Carrigy (1972) determined that feldspar and volcanic rock fragments constitute up to 40% of the detrital material in his samples while those from the Willow Creek Formation only once exceeded 15%. This in addition to the fact that no radiometric anomalies or uranium occurences are known to occur in the Willow Creek Formation indicates that it currently has a lower uranium resource potential (see Table 12.).

Table 11. Mineralization characteristics of the St. Mary and Blood Reserve Fms. and the Powder River Basin mining district.

	St. Mary-Blood Reserve Fms.		Powder RIver Basin	
	Description	Value	Description	Value
Deposit/ Occurrence	None	0	Mineralization (M)	10
			Outcrop rad. (O)	5
			U geochem. (G) anomaly	3
			Trace metals (T)	2
Host Lithology	Clastic Rocks (CQVF)	5	Clastic Rocks (CQVF)	5
	Low-rank (L) coal	3	Low-rank (L) coal	3
	Volcanics (FT)	3		
Uranium Provenance	Volcanic (V)	2	Volcanic (V)	2
Depositional Environment	Continental (CDL)	3	Continental (CDF)	5
	Marine (L)	2		
Structure	Facies (B) boundary	2	Facies (B) boundary	2
	Unconformity (U)	2		
Reductants	Plant debris (C)	2	Plant debris (C)	2
			Sulfide min. (S)	2
Permeability	Good (G)	3	Good (G)	3
Alteration	unknown	0	Bleaching (B) Limonite (L)	2

Total	27	46

Table 12. Mineralization characteristics of the Willow Creek Formation and the Powder River Basin mining district.

	Willow Creek Formation		Powder River Basin	
	Description	Value	Description	Value
Deposit/ Occurrence	None	0	Mineralization (M) Outcrop rad. (O) U geochem. (G) anomaly Trace metals (T)	10 5 3 2
Host Lithology	Clastic Rocks (CQVF) Low-rank (L) coal	5 3	Clastic Rocks (CQVF) Low-rank (L) coal	5 3
Uranium Provenance	Volcanic (V)	1	Volcanic (V)	2
Depositional Environment	Continental (CLD)	3	Continental (CDF)	5
Structure	Facies (B) boundary	2	Facies (B) boundary	2
Reductants	Plant debris (C)	2	Plant debris (C) Sulfide min. (S)	2 2
Permeability	Good (G)	3	Good (G)	3
Alteration	unknown	0	Bleaching (B) Limonite (L)	2
Total		19		46

The Cypress Hills

The Cypress Hills are composed of a series of Upper Cretaceous and Tertiary continental sandstones with tuffaceous, bentonitic and lignite horizons. Similar lithologies have been observed in uranium-bearing formations in Wyoming and South Dakota which stratigraphically correlate with the units in the Cypress Hills. The Ravenscrag Formation in the Cypress Hills correlates with the Fort Union Formation, and the Cypress Hills Formation correlates with part of the uraniferous White River Group. It would therefore appear that these formations exhibit some very favorable uranium-hosting characteristics. This fact has been previously recognized as the Cypress Hills has undergone as detailed an evaluation of its uranium potentail as has been undertaken in the Province.

Cameron et al. (1970) undertook a study of the radioactivity of coals in Western Canada, including the Cypress Hills. Coal samples around Eastend Saskatchewan exhibited the highest uranium concentration, as high as 825 ppm U Coal occurrences examined in Alberta (around Thelma) contained relatively low radioactivity. Nine of eleven sites showed 0-5 uR/hr, while the other two fell in the range of 6 to 25 uR/hr. This compares unfavorably with those aforementioned samples collected around Eastend which produced radioactivity levels greater than 100 uR/hr.

Later, Dyck et al. (1976) undertook a well-water reconnaissance in the Saskatchewan section of the Cypress

Hills. From 940 well samples and 60 spring water samples, U concentrations of 0.00 to 240.00 ppb, with a background value of 0.5 ppb were observed, in addition to a ^{222}Rn concentration range of 0.00 to 4135.00 pc/l and background reading of 250 pc/l.

Although some significant anomalies were detected, Dyck's results tend to negatively affect the uranium resource potential. These are:

1. The Cypress Hills Formation is the primary source of uranium in the samples. Lesser amounts of uranium are contributed from leached uranium-bearing lignites of the Ravenscrag Formation.
2. Deeper wells are lower in U and Rn concentrations.
3. No significantly large occurrences were detected.

Dyck's interpretation of these facts was that uranium is leached from the uppermost formations (areas of groundwater recharge) and subsequently dispersed laterally and vertically in underlying units.

Because of these results and the relatively small area of the region, the uranium resource potential of the Cypress Hills is considered low.

The Porcupine Hills Formation

The Porcupine Hills Formation is a Paleocene continental sandstone overlying the Willow Creek and Paskapoo Formations adjacent to the Foothills from Pincher Creek north to Olds–Sundre. Thus the formation lies within both the Oldman–Milk River drainage basin and the Bow–Red River drainage basin. The formation forms an asymmetrical north–south trending syncline, whose west limb dips

moderately (approximently 25°) to the east and the east limb that has a shallow dip to the west (Carrigy 1972). Structural evidence indicates that an erosional contact exists between the Porcupine Hills Formation and its underlying unit. Carrigy (1972) also shows that the paleocurrent direction and bulk composition differ with that of the Willow Creek and Paskapoo Formations. Sandstones of the Porcupine Hills appear to have been deposited in well-defined fluvial channels in and around numerous shallow lakes.

Detrital quartz, chert, nonvolcanic rock fragments and clastic carbonates are the major constituents of the sandstones. Both feldspar and volcanic rock fragments are rare. Carrigy (1972) ranks the abundance of source material as shown below:

sedimentary>>metamorphic>>volcanic

It appears that the primary source of clastic material in the Porcupine Hills Formation was the Paleozoic sedimentary units uplifted during early stages of the Laramide orogeny.

The uranium resource potential of the Porcupine Hills Formation must be fairly low as it lacks any apparent uranium source-material and is lacking in reductants such as plant remains or coal.

B. Rocky Mountains and Foothills

None of the formations in the Rocky Mountains and Foothills have been quantitatively assessed for their uranium potential during this study. The extrapolation of conditions defining the uranium mineralization potentials of the areas is inappropriate in regions of such geological complexity. Additionally, defining a resource model area from the United States which can be satisfactorily compared with this evaluation district would be highly speculative. Both difficulties are largely due to the presently unknown affects of the structural disturbance of the area upon processes of deposit genesis. At best this study is able to identify a number of circumstances which may reflect the area's potential for containing significant uranium mineralizaton.

As previously reported (under "Source Rocks") the Main Ranges of the Rocky Mountains contain some formations which may be a source of uranium. The most significant of these occurrences is in the Belt Supergroup in Southwest Alberta. where uranium mineralization is said to occur along with stratiform copper mineralization.

The Banff and Exshaw Formations were also evaluated as potential uranium source-rocks. Hydrogeology reports by Ozaray et al. (1977) in the Calgary-Golden area and Barnes (1977) in the Brazeau-Canoe River region would suggest that the permeability of these formations is sufficient to allow the movement of considerable volumes of groundwater.

In the foothills, both the Lower Cretaceous Blairmore and Upper Cretaceous Brazeau Group clastic wedges warrant consideration for hosting uranium. Mellon (1967) describes the stratigraphy and petrology of the Blairmore Group. Of particular significance with regard to the uranium geology of these units is:

1. The group is both feldspathic, and tuffaceous (the Crowsnest Volcanics occur in the Upper Blairmore Group), therefore a potential source-rock has been identified.
2. Plant material exists, and is most abundant in west-central Alberta in the 'Luscar facies'.
3. Lateral facies variations, which in a south to north direction change from nonmarine to marine units.
4. The existence in southwestern Alberta of a "red bed" facies which Mellon describes as being sub-areally formed.
5. An unconformity between the Lower Blairmore Group and (Cadomin Conglomerate) and the underlying Kootenay Formation, plus the local unconformity between the Upper Blairmore Mill Creek Formation and Middle Blairmore Beaver Mines Formation.

The significance of the "red beds" is unclear. They might constitute the type of "altered zones" such as those associated with "Wyoming-roll deposits. The "red-bed facies is likely to be generally deficient in uranium and mineralization is likely to occur only in carbonaceous pockets in the zone (as with peneconcordant deposits in Colorado) or where the zone transforms into a reducing facies.

The significance of unconformities is in their key role in developing "bifacies" type uranium deposits, whereby uraniferous groundwaters moving along an unconformity come in contact with an underlying permeable uranium reducing environment. Therefore this type of deposit may have

developed below either unconformity.

Additionally, Morton (pers. comm.) reports the possible occurence of uranium in a coarse clastic unit underlying a coal horizon in the Commotion Formation in the Kakwa River region.

The Brazeau Formation is stratigraphically and lithologically equivalent to the Paskapoo Formation. As such it would have a similar uranium resource potential. The only major factor differentiating these formations is how the structural disturbance in the Foothills affects the former and present-day groundwater geology. The following factors are likely to be of importance:

1. Groundwater permeability is decreased by increased cementation of the unit.
2. Fracturing the rock increases the vertical component of groundwater movement, as does the increased relief and formation dips.
3. Faulting can behave either as a conduit or barrier to groundwater movement.

The end result is a more variable groundwater pattern, both in terms of flow patterns and hydrochemistry. This latter factor stems from the fact that greater mixing of shallow and deep groundwater flow regimes occurs (Ozoray et. al., 1977, Barnes, 1976 and 1977). In this structurally disturbed setting, upward migrating hydrogen sulfide gas derived from underlying gas traps appears to also constitute a feasible mechanism for the reduction of any uranium in groundwaters.

C. Saskatchewan-Red Deer River drainage basin

In this region the Paskapoo Formation and the Edmonton Group represent the two most favorable uranium-hosting environments. The Belly River Group in east-central Alberta is not considered a likely uranium-host due to the uncertainty of a uranium-source and to the larger marine sediment component in the formation in this area. Table 13 summarizes the quantified uranium resources of this formation.

Table 13. Summation of estimated U^3O^8 resources in the Saskatchewan-Red Deer River drainage basin.

Formations/ Groups	Min. Ratio	Area U.S. (sq. mi.)	Area Fm. (sq. mi.)	U.S. Res. (s. tons)	Est. Res. (s. tons)
Edmonton	29/46	12,000	20,300	188,200	200,000
Paskapoo	25/46	12,000	18,800	188,200	160,000
Total					360,000

The Edmonton Group

The uranium resource potential of the Edmonton Group is promising primarily on the basis of the proximity of a uranium source-rock to a favorable host-environment. The sedimentary environment of depostion of these units closely resembles that of some uranium-bearing sandstones in the United States.

The three formations comprising the Edmonton Group are the Battle, the Whitemud and the Horseshoe Canyon

Formations. The Battle Formation is primarily composed of bentonitic shales and mudstones, in addition to the Kneehills Tuff. The formation may either host uranium deposits like the Catahoulla Tuff in Texas, or supply uranium to both overlying and to underlying strata. The fact that more sedimentary-hosted uranium deposits in the United States are in coarser units (Byers, 1978) and that the alteration of much of the volcanic detritus into bentonitic clays has occurred favors the latter suggestion. Gibson (1976) suggests that the paleoenvironment of the Battle Formation was a series of interconnected fresh-water lakes, and that the tuff is a diagenetically altered volcanic ash. Under these circumstances uranium would likely be released. Gibson also suggests that a relatively dry climate existed at this time which would favor the downward incursion of groundwater.

Beneath the Battle Formation, both the Whitemud and Horseshoe Canyon Formations exhibit features favorable for hosting uranium. These units contain abundant plant debris and coal and thus would be capable of reducing the uranium in groundwaters. Additionally, the depositional environment of an alluvial plain and prograding river delta is one which provides many of the preffered facies variations which control groundwater movement and develop uranium deposits.

Evidence of the potential of these formations is the fact that an airborne radiometric anomaly has been detected (Geol.Surv.Can.) in the Stettler-Buffalo Lake area (Township

39, Ranges 20, 21 and 22W4). Factors indicating the favorability of the Edmonton Group for containing uranium mineralization are summarized in table 14.

The Paskapoo Formation

The Paskapoo Formation is a successive continuation of the same clastic wedge which developed the Edmonton Group. Local unconformities developed between the Battle Formation and the overlying Paskapoo Formation (Scollard Member) as a result of changing sedimentation patterns which produced local areas of emergence. In some areas (the Drumheller district for example), subsequent fluvial sedimentation of the Paskapoo Formation produced sand-filled fluvial channels which downcut into the Battle and Whitemud Formations. Secondly, these channels may play an important role in uranium deposit genesis in two ways. Firstly, where the channels ar filled with plant debris, deposits could form if uranium was contributed from adjacent bentonitic and tuffaceous zones in the Battle Formation or from bentonite zones in the Scollard Member. The horizontal movement of groundwater along the unconformity could subsequently develop a "monofacies-type" deposit, where downward groundwater movement into a permeable reducing sediment occurs. Both models are dependent upon a horizontal groundwater flow pattern at the base of the Paskapoo Formation in the fluvial channels. The possibility of such occurrences is encouraging as fluvial channels are commonly

Table 14. Mineralization characteristics of the Edmonton Group and the Powder River Basin mining district.

	Edmonton Group		Powder River Basin	
	Description	Value	Description	Value
Deposit/ Occurrence	None	0	Mineralization (M)	10
	Outcrop rad. (O)	2	Outcrop rad. (O)	5
			U geochem. (G) anomaly	3
			Trace metals (T)	2
Host Lithology	Clastic Rocks (CQVF)	5	Clastic Rocks (CQVF)	5
	Low-rank (L) coal	3	Low-rank (L) coal	3
	Volcanics (FT)	3		
Uranium Provenance	Volcanic (V)	2	Volcanic (V)	2
Depositional Environment	Continental (CD)	3	Continental (CD)	5
	Marine (L)	2		
Structure	Facies (B) boundary	2	Facies (B) boundary	2
	Unconformity (U)	2		
Reductants	Plant debris (C)	2	Plant debris (C)	2
			Sulfide min. (S)	2
Permeability	Good (G)	3	Good (G)	3
Alteration	unknown	0	Bleaching (B)	2
			Limonite (L)	
Total		29		46

filled with coarser, more permeable, sediments than the surrounding rock facies. Additionally, the surrounding and underlying shaly, bentonitic zones of the Battle Formation would likely behave as an aquitard. Supportive evidence of these conclusions is suggested by the presence of groundwater springs on the Battle–Paskapoo contact (Le Breton et al., 1970).

Higher in the Paskapoo Formation, the importance of deriving other sources of uranium increases if uranium deposits are to develop. Therefore volcanic rock fragments and occasional bentonitic zones would be the most likely source of uranium. Carrigy (1971), suggests that the Paskapoo Formation represents part of a "northern facies" of sediment rich in volcanic and metamorphic material. This interpretation is supported by the abundance and variety of euhedral nonopaque heavy minerals, including hornblende, biotite, epidote, apatite and zircon.

Whether sufficient uranium occurs in the middle and upper sections of the Paskapoo Formation to permit formation of uranium deposits is unknown, but evidence to the affirmative as indicated by the occurrence of two airborne radiometric anomalies in the Paskapoo Formation (Geol. Surv. Canada data). These occur on the southwest shore of Sylvan Lake (Township 39, Range 2W5) and 15 miles northwest of Rocky Mountain House (Township 41, Ranges 8 and 9W5). Both anomalies cover an area of approximently one township.

The uranium resource characteristics of the Paskapoo Formation are given in table 15. As the formations was deposited in a dominantly fluvial-plain environment, the Powder River Basin is suggested as the most analagous U.S. mining district in which the uranium resource potential is estimated.

Table 15. Mineralization characteristics of the Paskapoo Formation and the Powder River Basin mining district.

	Paskapoo Fm.		Powder River Basin	
	Description	Value	Description	Value
Deposit/ Occurrence	None	0	Mineralization (M)	10
	Outcrop rad. (O)	2	Outcrop rad. (O)	5
			U geochem. (G) anomaly	3
			Trace metals (T)	2
Host Lithology	Clastic Rocks (CQVF)	5	Clastic Rocks (CQVF)	5
	Low-rank (L) coal	3	Low-rank (L) coal	3
	Volcanics (FT)	3		
Uranium Provenance	Volcanic (V)	2	Volcanic (V)	2
Depositional Environment	Continental (D)	3	Continental (CD)	5
Structure	Facies (B) boundary	2	Facies (B) boundary	2
Reductants	Plant debris (C)	2	Plant debris (C)	2
			Sulfide min. (S)	2
Permeability	Good (G)	3	Good (G)	3
Alteration	unknown	0	Bleaching (B) Limonite (L)	2
Total		17		46

D. The Athabasca River drainage basin

Two sectors of the region contain continental and nearshore clastic sediments in which uranium deposits could

possibly develop. In the west, extensions of the Paskapoo Formation and Edmonton Group equivalent (the Wapiti Group) are found. Their uranium-hosting potentials are the same as those noted in the Saskatchewan-Red Deer River basin (these units are described in detail in the discussion of the Saskatchewan-Red Deer River drainage basins). The other area worthy of consideration is in Northeast Alberta, particularly the McMurray Formation. The resource potentials of all three units is tabulated in table 16.

Table 16. Summation of estimated $U^3 0^8$ resources in the Athabasca River drainage basin.

Formations/ Groups	Min. Ratio	Area U.S. (sq. mi.)	Area Fm. (sq. mi.)	U.S. Res. (s. tons)	Est. Res. (s. tons)
McMurray	29/53	12,000	12,400	181,900	100,000
Wapiti	27/46	12,000	6,600	188,200	60,000
Paskapoo	23/46	12,000	10,400	188,200	80,000
Total					240,000

<u>The McMurray Formation</u>

The McMurray Formation is an Upper Cretaceous unit of the Lower Mannville Group, which constitutes one of the most favorable potential uranium-bearing formations in the Province. In addition, due to work done on the Athabasca Tar Sands, this formation has undergone some detailed investigations. A quantitative evaluation of the uranium resource potential of the McMurray Formation is therefore possible (see table 17). For quantification purposes the

McMurray Formation is compared with the Texas South Coastal Plain. This mining district shares a common type of paleoenvironment and diverse collection of potential uranium—reductants. Some significant differences do exist though, namely, that a different source—rock is envisaged and that currently there is no evidence that hydrogen sulfide has been a major reductant of uranium in the McMurray Formation.

Carrigy (1973), has divided the McMurray Formation into four informal units. From the base to the top these units are:

1. Pre—McMurray (?) beds have been recognized in isolated remnants consisting of a coarse— grained quartzose sandstone cemented by silica and goethite, devoid of bitumen and lying unconformably below the McMurray Formation.
2. A lower McMurray unit consisting of conglomerates, sands and shales deposited in lenticular beds. The thickest sections occur on paleosurface lows.
3. A middle McMurray unit is the primary bitumen—bearing sand with interbedded silts, shales and clays. These sands were deposited in a deltaic—esturine environment and exhibit a wide lateral and vertical variation of facies.
4. An upper McMurray unit is composed of marine shales, siltstones and cherty sandstones. Bedding is more commonly horizontal in nature.

A number of lithological characteristics exhibited by the McMurray Formation, particularly the lower and middle units, enhance its potential for hosting uranium. These include the presence of potential uranium reductants such as: lignite coal, framboidal pyrite fossilized plant remains and possibly anaerobic bacteria within and around the tar sands. Hematitic staining has been observed in recovered drill—core

Table 17. Mineralization characteristics of the McMurray Fm. and the South Texas Coastal Plain mining district.

	McMurray Fm.		Texas	
	Description	Value	Description	Value
Deposit/ Occurrence	None	0	Mineralization (M)	10
			Outcrop rad. (O)	5
			U geochem. (G) anomaly	3
	Trace metals (T)	1	Trace metals (T)	2
Host Lithology	Clastic Rocks (Q)	2	Clastic Rocks (CQVF)	5
	Low-rank (L) coal	3	Low-rank (L) coal	3
			Volcanics (FT)	3
Uranium Provenance	Granitic (G)	1	Volcanic (V)	2
Depositional Environment	Continental (CD)	5	Continental (CD)	5
	Marine (L)	2	Marine (L)	2
Structure	Facies (B) boundary	2	Facies (B) boundary	2
	Fold (Z)	1	Fold (Z)	1
	Unconformity (U)	2	Unconformity (U)	2
	Fracture (F)	1	Fracture (F)	1
Reductants	Plant debris (C)	2	Plant debris (C)	2
	Sulfide min. (S)	2	Sulfide min. (S)	2
	Ashphaltite (A)	2	H_2S (H)	2
Permeability	Good (G)	3	Good (G)	3
Alteration	unknown	0	Bleaching (B)	2
			Limonite (L)	

Total	29	57

in the lower unit of the McMurray Formation. The extent of this type of staining is unknown, but it could be part of an "altered zone" within the formation.

A number of structural features found in the McMurray Formation could also positively influence its uranium resource potential. The Devonian-Cretaceous unconformity whas been developed and altered by three general processes:

1. The Devonian karst topography.
2. Salt collapse structures in the underlying Elk Point Supergroup which appears to have developed vertical faulting in the McMurray Formation.
3. The paleodrainage pattern of the Devonian surface.

These features have produced an unconformity with a moderate relief, which in turn has a pronounced affect upon the groundwater patterns of the region. The primary aquifer of the formation is the lower unit. Salinity concentrations are highly variable, with formation waters moving in both directions across the unconformity (Gorrell et. al., 1975). Springs with high sulfate-ion concentration along the Clearwater River likely derive their water from underlying Devonian formations.

A number of types of uranium deposits could possibly occur in such an environment. "Monofacies-type" uranium deposits might develop just below the unconformity, in shaly zones, particularly if they have been faulted to permit

groundwater incursion. The paleokarst topography also could provide restricted low-lying zones in which plant material could accumulate and consequently remove uranium from groundwaters. The mixing of groundwaters vertically and laterally might also produce groundwater facies boundaries along which uranium could precipitate.

Groundwater aquifers of a relatively thin, and limited areal extent have been observed in the Tar Sands. Although the permeability in these aquifers is likely quite low, they do present zones of restricted groundwater flow, favorable for uranium precipitation. This is even more significant if the previously mentioned anaerobic bacteria are present in these zones.

The Grand Rapids Formation

A second younger thin clastic wedge of sediments above the McMurray Formation developed the Grand Rapids Formation. This unit is characterized fine-grained quartzose and feldspathic sandstone with interbedded siltstone, shale and small coal seams (Carrigy, 1973). The paleoenvironment of deposition of this formation is a shoreline-deltaic complex with the continental clastic material transforming into marine sedimentary units further to the west.

Two airborne radiometric anomalies have been detected in Townships 101 and 102; Ranges 11 and 12 (Geol. Surv. Canada data). These anomalies lie in an area where the Grand Rapids Formation and Clearwater Formation are exposed on the northeast side of the Birch Mountains Upland. Ozaray et. al.

(1978) have characterized this general area as one of groundwater recharge, with a subsequent downward direction of groundwater flow through the Grand Rapids Formation. As these anomalies lie on the east side of the Birch Mountains Upland, adjacent to the Clearwater River, it in fact may be a local area of discharge, as springs do occur along the river. Further work of this area and the formation as a whole is needed before the uranium resource potential of the area can be truly assessed.

E. The Peace River drainage basin

Three clastic wedges of continental sediments, derived from the west, occur in the sedimentary section in the Peace River drainage basin. Only the upper two wedges lie near enough to the surface to consider their uranium resource potential (Stott, 1972). The middle wedge is the Dunvegan Formation which overlies the marine shales of the Shaftesbury Formation. The upper wedge is the Wapiti Group which is found only in the southwest section of the evaluation district. Owing to its similarities with the Edmonton Group, particularly with regard to the presence of the the Kneehills Tuff and bentonitic zones, the favorability of the Wapiti Group for hosting uranium is believed to be approximently equal to that of the Edmonton Group. The only significant difference between these two units, in terms of their favorability characteristics, is that the Wapiti Group has no known radiometric anomalies.

This fact has been accounted for in calculating the 'mineralization ratio.' In the Peace River basin, the Wapiti Group is the only unit for which the uranium resource potential can be estimated (see table 18).

Table 18. Summation of estimated U^3O^8 resources in the Table 18. Summation of estimated U^3O^8 resources in the Peace River drainage basin.

Formations/ Groups	Min. Ratio	Area U.S. (sq. mi.)	Area Fm. (sq. mi.)	U.S. Res. (s. tons)	Est. Res. (s. tons)
Wapiti	27/46	12,000	5,000	188,200	50,000

The Dunvegan Formation

In Alberta the Dunvegan Formation is composed of fine-grained, calcareous and carbonaceous sandstone, interbedded marine- and nonmarine-shales. Its depositional environment is that of a delta front-marine interface (thicker continental sequences are found in northern British Columbia). Plant remains are relatively abundant. The uranium resource potential of this formation hinges upon whether a source of uranium exists to supply uranium to groundwaters passing through the Dunvegan Formation. The presence of a very large radiometric anomaly (Geol. Surv. Canada) in the Buffalo Head Hills area where the marine Shaftesbury Formation is the most Areally exposed formation suggests that it might be a potential host- or source-rock of uranium. The significance of either formation and of the radiometric anomaly are unclear.

The Wapiti Group

Stott (1972) states that the Wapiti Group is comprised of strata equivalent to the Belly River Group and the Edmonton Group, including the Scollard Member. This latter unit is included in the Wapiti Group as the Kneehills Tuff and the bentonites of the Battle Formation are a poor stratigraphic marker west of the Smoky River. The Ardley coal zone and the equivalent coal horizons are more suitable as stratigraphic markers at the top of the Wapiti Group. The Wapiti Group is lithologically comparable to the Edmonton Group, being composed of fluviatile, bentonitic sandstones with intermittent conglomerates, siltstones, mudstones, and coal seams (see table 19).

F. The Hay River basin

Deltaic and shallow—marine sandstones of the Upper Cretaceous Dunvegan Formation and the predominantly marine sandstones of the Bad Heart Formation appear to be the only significant clastic formations in the Hay River basin. The uranium resource potentials of both these formations in this basin appear to be relatively low, as no obvious source of uranium is found in the area. Both formations contain plant remains and thin coaly horizons which could serve to capture uranium from groundwaters.

Table 19. Mineralization characteristics of the Wapiti Group and the Powder River Basin mining district.

	Wapiti Group		Powder River Basin	
	Description	Value	Description	Value
Deposit/ Occurrence	None	0	Mineralization (M)	10
			Outcrop rad. (O)	5
			U geochem. (G) anomaly	3
			Trace metals (T)	2
Host Lithology	Clastic Rocks (CQVF)	5	Clastic Rocks (CQVF)	5
	Low-rank (L) coal	3	Low-rank (L) coal	3
	Volcanics (FT)	3		
Uranium Provenance	Volcanic (V)	2	Volcanic (V)	2
Depositional	Continental (CDL)	3	Continental (CDF)	5
Environment	Marine (L)	2		
Structure	Facies (B) boundary	2	Facies (B) boundary	2
	Unconformity (U)	2		
Reductants	Plant debris (C)	2	Plant debris (C)	2
			Sulfide min. (S)	2
Permeability	Good (G)	3	Good (G)	3
Alteration	unknown	0	Bleaching (B)	2
			Limonite (L)	
Total		27		46

Conclusions

Two issues should be addressed in these concluding statements. Firstly, how effective were the various techniques adopted in this study in providing useful guidelines for assessing the uranium resource potential of sedimentary formations in the Province? Secondly, what significance should be given to the speculative uranium resources of Alberta calculated herein?

The evaluation techniques used in this study provide only indirect indications of the characteristics of hypothetical uranium mineralization in the Province. This is due to the fact that Alberta constitutes virgin territory for uranium exploration and therefore little information directly related to uranium resources exists. Lithological descriptions of potential uranium source- and host-formations generally lack the type of information needed to outline exactly which areas of any given unit are favorable or unfavorable for supplying or hosting uranium. This limitation is not considered to be important with regard to source-rocks, as they likely encompass a large portion, if not all, of the source formation. However favorable uranium-host environments generally constitute only a small part of the host formation. For instance, well evaluated regions of the United States, have often identified 'linear trends' favorable for hosting uranium mineralization. These trends may be the areal distribution

of an "alteration front" or a paleochannel system. Meyboom's (1960) facies variation maps of the Milk River Formation may be interpreted to roughly delineate the paleoshoreline environment at the time the sediments were deposited. This information, which generally concurs with the uranyl– and sulfate–ion trends in the formation, provide the basis from which the resource evaluation area of the Milk River Formation was delineated. The only other data of this kind are either too generalized or too limited in areal coverage to characterize in detail the potential uranium resources of any given formation. Mellon's (1967) facies interpretation of the Blairmore Group is helpful, but too generalized, to attempt a more detailed interpretation of the unit. Holter's (1975) sandstone/siltstone distribution maps are restricted to the Scollard Member in that district where it coincides with the Ardley coal zone, and are therefore too limited in terms of detailing possible favorable uranium host–environments in the lower units of the Paskapoo Formation.

The extrapolation of Breger's (1974) correlations of coal properties with their uranium contents to Alberta coals is analagous to using a 'pathfinder element' in a geochemical survey. There is no proof though that Breger's trends actually occur in Alberta coals. Thus the calculated potential uranium contents illustrated in figures 10 and 11 are unproven. Nonetheless this exercise does aid in identifying potentially uraniferous coals and serves to

define which regions should be given priority.

The gamma-log survey was of very limited applicability in recognizing any uraniferous provinces in Alberta strata. The technique described by Zeller et. al. (1976) would appear to work well in formations which are lithologically constant, relatively low in K^{40} content and with an abundant supply of high quality gamma-log information. If these conditions are not met, it is necessary to adopt very discriminating criteria which in all likelihood will omit more possible uranium anomalies than the method will identify. This is a result of the inability of gross-count scintillometers to differentiate uraniferous radioactive sources from unidentifed variations of drillhole conditions and from changes in the ^{40}K content (due to lithological variations in most circumstances). In this study, the homogeneity of lithology applies to relatively few formations and only in a very generalized manner. The quantity of good quality gamma-logs with logged sections in formations of interest for their uranium resource potential is barely adequate for this type of work. Therefore no uraniferous horizons were identified by this technique.

The ultra-violet fluorescence technique alone proved to be practical in analyzing the uranium concentrations of Albertan groundwaters. The pilot study using samples of Milk River Formation groundwaters indicates that the method constitutes a powerful exploration tool, particularly when it is combined with groundwater studies of other dissolved

ions. It should be remembered that any dissolved ion distribution pattern observed in groundwaters in any given formation are current distribution patterns only. Paleodistribution patterns may have exerted a stronger influence in controlling uranium mineralization. The current groundwater patterns exhibited by the McMurray Formation might have been radically different in the past, particularly if the heavy oil was derived from underlying formations. Therefore current groundwater patterns may not reflect the distribution of uranium mineralization in the McMurray Formation or other rock units which may have notably altered their groundwater patterns. Of all the techniques used in this study, this method provided the most direct indication of the behavior of uranium in a formation.

The economic evaluation of mining sandstone-uranium deposits using the PRICE2 computer program is based upon the best available data compiled from the U.S. uranium mining industry. Applying these data in an Albertan context introduces an error which although indeterminant, is probaly relatively constant in the range of circumstances examined. Two other unquantifiable errors affecting any interpretation of the results are variations of real from-predicted-future costs and uranium pricing. These errors are solely dependent upon projected cost indexes and the predicted 1990 contract-selling price of uranium. Although the future-selling price of uranium of $40 U.S./lb. U_3O_8 was based on very limited data, any error in this prediction

will not directly affect the computer-derived results as the PRICE2 program calculated the required selling-price of uranium over a wide range of prices. Thus it would seem that those errors in transferring U.S. mining data to an Albertan model and in forecasting cost escalation rates are the most significant.

Despite these potential errors, the PRICE2 results are current 'best estimates' of the economic conditions of mining sandstone-uranium deposits in Alberta beginning in 1990. In the interpretation of these results, it should be remembered that the boundaries separating economic from subeconomic conditions in the depth/thickness versus tonnage graphs (figures 22 to 24) are only guidelines. The relative economic favorabilities of underground mining versus open pit mining are probably much more accurate, as errors resulting from inaccurate index projections are equal in both instances. Consquently, the observation that the minimum economically exploitable tonnage in open-pit mining is less than that in underground mining is quite reliable (assuming suitable depth/thickness ratios). Assuming that small- and medium-tonnage deposits are more numerous than large deposits the potential for open-pit uranium mines to be developed in Alberta is greater than the potential for underground uranium mines. The potential for open-pit mines over underground mines is further enhanced, for shallow deposits are more likely to be found than deeper ones in early stages of exploration.

The calculated uranium resource estimates are inherently optimistic, for they assume that the uranium resources in the Alberta formations are proportional to the uranium resources in the U.S. uranium mining districts. These results are of the most speculative kind and might be erroneous by several orders of magnitude.

Systematic calculations of the mineralization favorability of the Upper Cretaceous and Tertiary formations indicate that they have similar potentials for hosting uranium, and that their calculated potential uranium resources are dominantly influenced by the areal coverage of near- to surface- outcrop. A more detailed evaluation would undoubtably differentiate those formations with a higher potential for hosting uranium from those with a lower potential. Other formations should not currently be quantified as a low mineralization favorability ratio would be derived due to a lack of data.

When these uranium resource estimates are considered in terms of the PRICE2 computer results, some interesting observations are possible. From the PRICE2 data it appears that the minimum deposit tonnage economically exploitable in Alberta when operating an open pit and underground mine are about 0.5 million tons and 4.0 million tons respectively (assuming an average ore grade of 0.10% U_3O_8). From such deposits, the recovered ore would total about 500 and 4,000 short tons of U_3O_8 respectively. In the Milk River Formation, which has a speculative uranium resource of 3,000

short tons, a maximum number of five open-pit mineable deposits are projected to exist. No underground mining operations would be economical in these condtions. This assumes all the resource exists in these deposits. In other formations, whose evaluation areas are considerably larger, the maximum number of potential deposits calculated from the resource estimate are proportionatly larger.

In simple qualitative terms, this study has indicated that continental and nearshore-marine clastic formations in Alberta exhibit a number of characteristics observed in uraniferous sandstones of the United States. These common characteristics are generally a result of common processes being active during sediment deposition and diagenesis. Most notably, both U.S. and Alberta formations exhibit similar lithological facies variations, are rich in plant remains and in most cases near a potential source-rock. The study also proves the necessity to further define which formations or parts thereof which are favorable for hosting uranium, in order to measure the uranium resource potential of the Province with higher accuracy. More detailed work, directly measuring the uranium distribution in all formations of interest and the determination of how these patterns are influenced by facies variations, would be most beneficial. Groundwater studies and field mapping are two techniques that should yield significant information.

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VII. Appendix 1 - Mineralization Characteristics

This table of mineralization characteristics has been designed with two purposes in mind. The first is to numerically rank these factors for calculating a mineralization ratio. The second is to list the characteristics in a manner compatible with tables developed by the National Uranium Resource Evaluation Preliminary Report (1976). These tables describe the relevant geological characteristics regarding uranium- bearing formations in the United States. To facilitate this comparison, each geological factor in this study has been classified under the same headings and given the same lettered symbols under these headings as used in the NURE report (not all headings from the NURE report are used in this study as it includes some types of deposits not likely to be found in Alberta). Some factors have not been numericlly ranked and are included to facilitate the comparison between formations described in the NURE report and formations in Alberta.

Table 20. Mineralization Characteristics Guidelines.

<u>Deposit/Occurrence Characteristics.</u>

10	Surficial uranium mineralization (M)
5	Anomalous radiation at outcrop (O)
3	Geochemical U anomaly (G)
2	Trace metals (T)

<u>Lithological and Occurrence Control.</u>

5	Clastic Rocks: Coarse (C), Fine (F), Arkosic (A), Calcareous (Ca), Feldspathic (Fs), Quartzose (Q), Volcanic (V)
	Other sedimentary rocks:
1	Carbonates (Ca)
1	Chert (C)
1	Evaporites (E)
3	Low-rank coal (L)
3	Phosphates (P)
3	Volcanics: Extrusive (E), Felsic (F), Mafic (M), Tuffaceous (T), Vitric (V)

<u>Uranium Provenance</u>

2	Granitic (G)
2	Volcanic (V)
1	Phosphates (P)
1	Organic Shales (S)

<u>Enviroment</u>

5	Continental: Paleochannel (C), Deltaic (D), Eolian (E), Fan (F), Lacustrine (L)
2	Marine: Abyssal (A), Bathyal (B), Littoral (L), Sabkha (S)

<u>Structure</u>

1	Breccia (B)
1	Contact (C)
1	Dissemination (D)
1	Fracture (F)
1	Pipe (P)
1	Vein (V)
2	Unconformity (U)
2	Facies boundary (B)
1	Fold (Z)

Reductants

2	Ashphaltite (A)
2	Hydrogen Sulfide (H)
2	Carbon/Coal/Plant fossil debris/Humates (C)
2	Natural Gas (G)
2	Sulfide Minerals (S)
1	Petroleum (P)

Permeability

3	Good (G)
2	Fair (F)
1	Poor (P)

Alteration

2	Bleaching (B), Chlorite (C), Hematite (H) Limomite (L), Sericite (S)

Cementation

2	Calcite (C), Hematite (H), Limonite (L), Phosphate (P), Silica (S)

VIII. Appendix 2 - Coal Calculations and Results

Steiner et.al. (1973), described the characteristics of Alberta coal in terms of the following parameters.*

1. Gross calorific value.
2. Fixed carbon content.
3. Moisture content.
4. Ash content.
5. Sulfur content.

Breger (1973) correlated the uranium concentratins of his coal samples with their dry, mineral -free calorific values and volatile matter content.

Converting data from Steiner et.al. (1973), to their apparent uranium content, required that the gross calorific values and fixed carbon contents of Alberta coals be converted to a dry mineral-free calorific value and volatile matter content using Parr's formulas:

$$\text{Dry, Mn-free Btu} = \frac{\text{Btu} - 5S}{100 - (M + 1.08A + 0.55S)} \times 100$$

$$\text{Dry, Mn-free VM} = 100 - \text{Dry, Mn-free FC}$$

where,

Mn = mineral water
Btu = British thermal units per pound
FC = percentage of fixed carbon
VM = percnetage of volatile matter
M = percentage of moisture
A = percnetage of ash
S = percnetage of sulfur

These results are converted to an apparent uranium

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*Coal samples from the Belly River Formation were not included.

concentration using figures 8 and 9. Tables 21 and 22 are compilations of the factors used to derive the apparent uranium concentrations of coals from Alberta.

Table 21. Data utilized for calculating apparent uranium contents of Alberta coal from their volatile matter contents.

No. of Occurrences	Fixed Carbon	Volatile Matter	% U
1	50	50	0.19
2	52	49	0.28
1	53	47	0.34
1	55	45	0.48
3	56	44	0.6
10	57	43	0.7
19	58	42	0.8
17	59	41	1.0
6	60	40	1.1
3	61	39	1.3
4	62	38	1.7
2	63	37	2.0

Table 22. Potential uranium concentrations in Alberta coal estimated from their calorific values.

Gross Btu	Formation	% S	% Ash	% Moisture	Dry Btu	% U
7150	Ke	0.4	6	33	11,824	11
7400	Kfr	0.4	6	29	11,477	13
7802	Ke	0.4	6	27	11,720	10
7970	Ke	0.5	6	30	12,564	6.7
7998	Ke	0.4	6	25	11,680	11
8085	Ke	0.4	6	27	12,164	9.0
8155	Ke	0.4	6	28	12,419	8.0
8180	Tp	0.4	9	19	11,483	13
8270	Ke	0.4	8	26	12,655	7.0
8277	Ke	0.4	8	28	13,077	4.8
8280	Ke	0.4	7	28	13,082	4.8
8400	Ke	0.4	7	17	11,141	14
8430	Ke	0.4	8	27	13,112	4.8
8460	Ke	0.4	6	27	12,730	6.0
8547	Ke	0.3	8	21	12,155	9.0
8575	Ke	0.3	7	25	12,724	6.0
8590	Ko	0.4	6	26	12,734	6.0
8635	Ke	0.3	8	20	12,107	9.0
8648	Ke	0.4	8	25	13,045	5.0
8667	Kfm	0.5	8	23	12,692	6.0
8670	Ke	0.4	6	26	12,828	5.5
8680	Ke	0.4	7	25	12,883	5.5
8680	Ke	0.4	7	25	12,883	6.0
8715	Ke	0.5	8	21	12,400	7.5
8753	Ke	0.3	7	25	12,988	5.0
8755	Ke	0.4	8	20	12,279	8.5
8785	Ke	0.2	8	18	12,315	9.5
8832	Ke	0.5	8	20	12,389	7.5
8850	Ke	0.4	7	26	13,334	3.7
9825	Ke	0.5	8	21	12,699	7.0
9860	Kfm	0.5	8	19	12,396	7.5
9050	Ke	0.5	10	18	12,739	6.0
9065	Ke	0.3	8	21	12,893	5.5
9120	Ke	0.5	8	23	13,355	3.7
9220	Ke	0.5	8	23	13,502	3.0
9270	Ke	0.7	9	19	13,025	4.8
9300	Ke	0.3	10	17	12,890	5.5
9368	Ke	0.4	8	20	13,140	4.5
9387	Ke	0.3	10	18	13,194	4.3
9447	Ke	0.3	9	18	13,080	4.8
9455	Ke	0.4	9	19	13,278	3.8
9480	Ke	0.3	10	17	13,140	4.5
9490	Kfm	0.7	8	20	13,320	3.7
9497	Ke	0.3	8	20	13,319	3.7

Gross Btu	Formation	% S	% Ash	% Moisture	Dry Btu	% U
9524	Ke	0.3	8	19	13,172	4.5
9573	Ke	0.7	10	13	12,580	7.0
9575	Ko	0.8	10	12	12,422	7.8
9610	Ke	0.5	10	18	13,514	3.0
9715	Ke	0.3	10	16	13,281	4.0
9815	Kfm	0.5	8	18	13,395	3.6
9910	Kfm	1.0	10	16	13,571	2.4
10300	Kwt	0.4	8	18	14,096	1.4
10600	Kwt	0.4	8	11	13,201	4.3
10645	Kwt	0.3	10	12	13,798	2.1
10690	Ko	0.6	10	12	13,867	2.0
10750	Ke	0.6	10	11	13,766	2.1
10753	Kwt	0.4	10	11	13,764	2.1
10798	Kwt	0.2	10	10	13,640	2.7
11080	Kwt	0.5	10	8	13,661	2.7
11465	Kwt	0.4	8	12	14,461	0.8
11265	Kwt	0.4	10	9	14,059	1.5
11295	Kwt	0.6	12	6	13,975	2.0

Ke – Edmonton Group

Kfm – Frenchman Formation

Ko – Oldman Formation

Kwt – Wapiti Formation

Tp – Paskapoo Formation

IX. Appendix 3 – Fission-track analysis experimental technique

A. Sample Preparation

The Lexan plastic is cut into 1 cm by 0.5 cm pieces and cleaned. For best results, ultrasonic water-bath cleaning of the plastic chips in a neutral, aqueous soap solution was performed. The chips were then rinsed with dilute hydrochloric acid and distilled water. These were then deposited in the plastic "rabbit" vials, each containing 1 ml of sample solution (no pre-treatment of the water sample is necessary). The vial would then be one-half to three-quarters full. Filling the vial completely would result in leakage when the vial was sealed by melting the lip of the vial top with a soldering iron.

B. Sample irradiation

Irradiation periods ranged from 1000 to 3000 seconds, under a neutron flux of 10^{12} neutrons/cm^2. A cooling-off period of 2 to 3 days is necessary to allow the radioactivity of the samples to fall to safe background levels. As a precautionary measure, all samples were subsequently handled using plastic gloves.

C. Fission-track Etching and Counting

To prepare the fission-tracks for viewing with a microscope the plastic chips were etched in 6.5N NaOH solution. Optimum results were achieved by etching the chips in a dry-bath at 65 °C for 40 minutes. The plastic chips were then cleaned and dried for fission-track counting.

Counting the fission-tracks was done at 250X magnification with an approximate viewing area of 1 mm by 0.5 mm.

X. Appendix 4 - Uranium analysis by ultra-violet fluorescence: experimental procedure

Prior to sample analysis, two standard solutions with concentration of uranium of the same order of magnitude as anticipated in the unknown samples, are prepared. For this study, solutions of 5 ppb and 1 ppb U were prepared on a daily basis from a 1 ppm standard solution kindly provided by Noranda Exploration Ltd. Six milliliters of a standard and sample solution were pipetted into separate test cells. In each of these cells, 0.8 ml. of buffer solution were added.

The uranly-ion concentration in solution was determined by measuring intensity of the laser-induced fluorescence which is directly proportional to its uranium concentration. Alternately, the test solution vial and sample solution vial were inserted into the analyser until consistant results were produced (3 to 5 times for each vial in most instances). The ratio of the sample solution-reading to the standard solution-reading multiplied by the U concentration of the standard solution equals the measured U content of the unknown solution.

XI. Appendix 5 - The PRICE2 program

When PRICE2 commences operation, a series of briefly annotated questions will appear on the terminal requesting data for each input parameter. These parameters are described in this appendix under four general groups which are:

1. General parameters.
2. Cost parameters.
3. Physical parameters (open pit and underground).
4. Time frame parameters.

A. General Parameters

This first group of parameters is generalized information which will apply for both open pit and for underground mining operations. The program allows successive runs keeping the data in this section constant and varying the other parameter groups.

1. DATEC - the dates at which cost inputs were taken.
2. DATEP - the date for which the price output data are valid. For most cases this date is when uranium oxide (yellowcake) production begins.
3. PARAM(1) - the Marshall and Swift mining and milling equipment cost index (M&S Index) at DATEC. This index is published on a monthly basis in the Chemical Engineering Magazine and is a purchasing cost index of U.S. mining and milling equipment.
4. PARAM(2) - the M&S index at DATEP. See figure 10 for the projected M&S indexes to the year 1995.
5. PARAM(3) - the general wholesale price index at DATEC published by Statistics Canada (Canada Year Book, 1978 used here).
6. PARAM(4) - the WP Index at DATEP. See figure 11 for projected wholesale price indexes to the year 1995.
7. PARAM(5) - Royalty (percentage of mine mouth ore value).
8. PARAM(6) - percentage depletion allowance.*

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*33.3% is the basic allowance (Stikeman, 1978) but it will vary slightly in each individual case.

9. PARAM(7) – effective tax rate (combined federal and provincial taxes).
10. PARAM(8) – investment tax credit. *
11. PARAM(9) – the percentage of mill cost qualifying for investment tax credit.
12. PARAM(10) – the D/T ratio of field expense and drilling cost input.
13. PARAM(11) – the maximum price in dollars per pound uranium oxide calculated by PRICE2. Klemenic(1974) provides the best information of this type for a D/T ratio of 56:1. See Appendix A in Ellis(1979).
14. PARAM(12) – the increments used in calculating the minimum price(cents per pound) to obtain a given rate of return. This is done by comparing the differance between the present value of the investment cash flow (PVINV) and present value of the net income cash flow (PVCF) at a given price. Interpolation is then used to recalculate the selling price until the PVINV equals the PVCF. As interpolation is used, a PARAM(12) value of 50. is recommended.
15. MILDEP – mill depreciation life in years.
16. FLAG(1) – a positive integer for straight-line depreciation; and a negative integer for double declining depreciation switching to straight line depreciation when beneficial.
17. ROR – required discount cash flow rate of return (percent). Five values can be entered in ascending order, followed by a negative flag.
18. AVEGRD – an average ore grade, for which a corresponding percentage mill recovery is given in the next parameter array (this is not the ore grade of the deposit).
19. MILREC – the percentage mill recovery corresponding to the ore grades from AVEGRD.
20. ESCALN – escalation rate in percent. The first and second values define the revenue escalation rates in the first and second periods. The third and fourth parameters define the cost escalation rates in the first and second periods. These periods are defined by NYEAR(10,1) and NYEAR(10,2) respectively.

For this study, table 23 lists the generalized data values utilized.

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*This tax credit varies according to what types of capital expenditures are made. 5% of qualified capital expenditures is the basic deduction.

Table 23. Generalized parameter values inserted in PRICE2 for this study.

Parameter	Values	Parameter	Values
DATEC	Oct. 1979	AVEGRD (1)	0.01
DATEP	Jan. 1990	AVEGRD (2)	0.025
PARAM (1)	613.4	AVEGRD (3)	0.05
PARAM (2)	1100.	AVEGRD (4)	0.10
PARAM (3)	687.	AVEGRD (5)	0.20
PARAM (4)	1220.	AVEGRD (6)	0.25
PARAM (5)	12.5	MILREC (1)	47.5
PARAM (6)	33.3	MILREC (2)	77.5
PARAM (7)	51.	MILREC (3)	87.5
PARAM (8)	5.	MILREC (4)	92.5
PARAM (9)	70.	MILREC (5)	94.0
PARAM (10)	56.	MILREC (6)	95.0
PARAM (11)	100.	ESCALN (1)	3.00
PARAM (12)	50.	ESCALN (2)	5.00
MILDEP	5 or 10	ESCALN (3)	5.00
FLAG (1)	-1	ESCALN (4)	5.00
ROR (1)	15.		
ROR (2)	18.		
ROR (3)	21.		
ROR (4)	24.		
ROR (5)	27.		
ROR (6)	-1		

B. Cost Parameters

Cost data are now entered. Ellis (1979) defined all COSTPT variables in average dollars per short ton of ore produced during the mine life as these were the units used by Klemenic (1974). These cost entries are those which apply at DATEC, and are subsequently recalculated to DATEP using the M&S and WP price indexes. None of these factors are affected by the length of the production period. Consequently each parameter remains constant for both the 5— and 10—year mine life periods in their respective mining techniques.

1. FLAG(2) — for underground mining a negative number and for open pit mining a positive number.
2. TPD — daily mine production in short tons per calender year.
3. AVGRD — the average ore grade of the deposit in percent U_3O_8.
4. COSTPT(1) — Field expense.
5. COSTPT(2) — Property acquisition.
6. COSTPT(3) — Exploration drilling.
7. COSTPT(4) — Development drilling.
8. COSTPT(5) — Mine primary development.
9. COSTPT(6) — Mine plant and equipment, except those purchased during the mine primary development.
10. COSTPT(7) — Mill construction.
11. COSTPT(8) — Mining.
12. COSTPT(9) — Haulage.
13. COSTPT(10) — Milling.

Table 24. Cost data for open pit mines in Alberta with an average grade of 0.05% U_3O_8 and a 5- or 10-year production period.

Parameters	1000 TPD	2000 TPD	3000 TPD
COSTPT(1)	0.13	0.13	0.13
COSTPT(2)	0.10	0.10	0.10
COSTPT(3)	0.18	0.18	0.18
COSTPT(4)	0.06	0.06	0.06
COSTPT(5)	8.24	7.49	7.14
COSTPT(6)	0.28	0.27	0.25
COSTPT(7)	2.82	2.29	2.03
COSTPT(8)	1.19	1.04	0.97
COSTPT(9)	1.12	1.12	1.12
COSTPT(10)	7.60	6.02	5.24

Table 25. Cost data for open pit mines in Alberta with an average grade of 0.10% U_3O_8 and a 5- or 10-year production period.

Paremeters	500 TPD	1000 TPD	2000 TPD
COSTPT(1)	0.45	0.45	0.45
COSTPT(2)	0.36	0.36	0.36
COSTPT(3)	0.60	0.60	0.60
COSTPT(4)	0.18	0.18	0.18
COSTPT(5)	9.31	9.31	8.80
COSTPT(6)	0.31	0.31	0.30
COSTPT(7)	3.68	3.00	2.43
COSTPT(8)	2.68	2.31	2.09
COSTPT(9)	1.12	1.12	1.12
COSTPT(10)	9.82	7.78	6.29

Table 26. Cost data for underground mines in Alberta with an average ore grade of 0.05% U_3O_8 and a mine production period of 5- or 10-years.

Parameters	1000 TPD	2000 TPD	3000 TPD
COSTPT(1)	0.13	0.13	0.13
COSTPT(2)	0.10	0.10	0.10
COSTPT(3)	0.18	0.18	0.18
COSTPT(4)	0.06	0.06	0.06
COSTPT(5)	4.15	3.68	3.32
COSTPT(6)	0.75	0.66	0.56
COSTPT(7)	2.82	2.29	2.03
COSTPT(8)	7.00	6.26	5.81
COSTPT(9)	1.12	1.12	1.12
COSTPT(10)	7.60	6.02	5.24

Table 27. Cost data for underground mines in Alberta with an average ore grade of 0.10% U_3O_8 and a mine production period of 5- or 10-years.

Parameters	500 TPD	1000 TPD	2000 TPD
COSTPT(1)	0.45	0.45	0.45
COSTPT(2)	0.36	0.36	0.36
COSTPT(3)	0.60	0.60	0.60
COSTPT(4)	0.18	0.18	0.18
COSTPT(5)	6.40	6.40	5.46
COSTPT(6)	1.22	1.22	1.00
COSTPT(7)	3.68	3.00	2.43
COSTPT(8)	11.77	10.43	9.39
COSTPT(9)	1.12	1.12	1.12
COSTPT(10)	9.82	7.87	6.29

C. Physical Parameters

The PRICE2 computer program procedes further by requesting information pertinent to either open-pit or underground mining. In each case the variables defined mainly concern the ore depth to thickness (D/T) ratios to be evaluated and how costs increase as the D/T ratio increases. The following set of variables apply to open pit mining:

1. FLAG(3) – A negative or positive integer indicates the absence or presence of external taxable income respectively.
2. PITDAT(1) – The ore depth to thickness ratio for which cost input applies.
3. PITDAT(2) – The percentage increase in field expenses and drilling costs due to a doubled depth to thickness ratio.
4. PITDAT(3) – The percentage increase in mine primary development costs with a doubled depth to thickness ratio.
5. PITDAT(4) – The percent increase in backfilling costs with a doubled depth to thickness ratio.
6. PITDAT(5) – The percentage of the final pit to be backfilled.
7. PITDAT(6) – The backfilling costs in cents per cubic yard.
8. PITDAT(7) – Ratio of surrounding area to be reclaimed to the area of the pit surface.
9. PITDAT(8) – The pit surface reclamation costs after backfilling (dollars per acre).
10. PITDAT(9) – The surrounding area reclaimation costs (dollars per acre).
11. PITDAT(10) and PITDAT(11) – coefficients used to calculate the environmental impact study costs. Ellis (1979) assumed that these costs increased logarithmically from $150,000 for a 500 TPD open pit to $500,000 for a 10,000 TPD operation. With the coefficents provided, this expense is calculated for any size deposit based upon Ellis's assumption. The following formula was used:

    Cost = PITDAT(10) * ALOG(10) - PITDAT(11)

12. DTREQ – The ore depth to thickness ratios required. Up to 40 ratios can be entered, in ascending order. The last number must be 0.0.

Following are descriptions of the comparable variable used in underground mining operations:

1. FLAG(3) – A negative or positive integer indicates the absence or presence of external taxable income respectively.
2. UGDATA(1) – The ore depth to thickness ratio for which cost input applies.
3. UGDATA(2) – The percentage increase in field expenses and drilling costs due to a doubled depth to thickness ratio.
4. UGDATA(3) – The percentage increase in mine primary development costs with a doubled depth to thickness ratio.
5. UGDATA(4) – The percent increase in mining costs with a doubled depth to thickness ratio.
6. UGDATA(5) – The percentage increase in costs due to mine health and safety regulations.
7. UGDATA(6) and UGDATA(7) – coefficients used to calculate the environmental impact study costs. Ellis (1979) assumed that these costs increased logarithmically from $150,000 for a 500 TPD open pit to $500,000 for a 10,000 TPD operation. He further assumed that the environmental impact study costs for an underground deposit was 2/3 that of an open pit mine of the same size. With the coefficients provided, this expense is calculated for any size deposit based on Ellis's assumption. The following formula was used:

   Cost = PITDAT(10) * ALOG(10) - PITDAT(11)

8. UGDATA(8) – The percentage of reserves not recovered in mining, but were left as pillars.
9. DTREQ – The ore depth to thickness ratios required. Up to 40 ratios can be entered, in ascending order. The last number must be 0.0.

For both open pit and underground mining operations these variables remain constant for 5–year and 10–year mine production models. These variables are independent of any time–frame considerations.

Table 28. Parameters defining the physical characteristics of the open-pit and underground mines evaluated by PRICE2.

Parameters	Open-pit mining	Parameters	Underground mining
FLAG(1)	-1	FLAG(1)	1
PITDAT(1)	24.	UGDATA(1)	76.
PITDAT(2)	100.	UGDATA(2)	100.
PITDAT(3)	100.	UGDATA(3)	100.
PITDAT(4)	100.	UGDATA(4)	12.
PITDAT(5)	100.	UGDATA(5)	5.
PITDAT(6)	50.	UGDATA(6)	179345.
PITDAT(7)	2.	UGDATA(7)	384053.
PITDAT(8)	6000.	UGDATA(8)	25.
PITDAT(9)	3000.		
PITDAT(10)	269018.		
PITDAT(11)	576079.		
DTREQ(1)	10.	DTREQ(1)	10.
DTREQ(2)	20.	DTREQ(1)	20.
DTREQ(3)	30.	DTREQ(3)	30.
DTREQ(4)	40.	DTREQ(4)	40.
DTREQ(5)	50.	DTREQ(5)	50.
DTREQ(6)	60.	DTREQ(6)	60.
DTREQ(7)	70.	DTREQ(7)	70.
DTREQ(8)	80.	DTREQ(8)	80.
DTREQ(9)	0.	DTREQ(9)	90.
		DTREQ(10)	100.
		DTREQ(11)	120.
		DTREQ(12)	140.
		DTREQ(13)	0.

D. Time Frame Parameters

The last variables needed by PRICE2 defines the 'time frame' of the model. All NYEAR variables are two dimensional. Except for NYEAR(10,2), the first digit is the year in which the given task begins and the second digit is the year in which it ends.

1. NYEAR(1,2) – Field expense.
2. NYEAR(2,2) – Property acquisition.
3. NYEAR(3,2) – Exploration drilling.
4. NYEAR(4,2) – Development drilling.
5. NYEAR(5,2) – Mill construction.
6. NYEAR(6,2) – Operating costs
7. NYEAR(7,2) – Enviromental impact statement.

8. NYEAR (8,2) - Backfilling after mining ceases.
9. NYEAR (9,2) - Reclamation. The last year of reclamation is the final year of the project.
10. NYEAR (10,2) - the number of years until the new cost andprice escalation rates apply respectively.
11. YRPDV - Mine primary development. Values are real numbers, therefore 8.0 is the beginning of year 8.
12. MPEQYR - Plant and equipment purchase dates ranked from earliest to latest. A negative integer must follow these dates. None of these purchase dates occurs prior to mine primary development.
13. NENV - The number of years from 1977 to the year in which the environmental impact study is undertaken. For each of these years, the cost of such a study calculated from Ellis's 1977 model are increased 10% per annum.

Table 29. Time from parameters for open-pit and underground mines with a production period of 5- and 10-years.

Parameters	Open-Pit (5-years)	Open-Pit (10-years)	Underground (5-years)	Underground (10-years)
NYEAR(1,2)	1,5	1,5	1,5	1,5
NYEAR(2,2)	1,5	1,5	1,5	1,5
NYEAR(3,2)	3,6	3,6	3,6	3,6
NYEAR(4,2)	5,7	5,7	5,7	5,7
NYEAR(5,2)	9,10	9,10	10,11	10,11
NYEAR(6,2)	11,15	11,20	12,16	12,21
NYEAR(7,2)	8,8	8,8	8,8	8,8
NYEAR(8,2)	16,16	21,21	-	-
NYEAR(9,2)	14,18	14,23	-	-
NYEAR(10,2)	10,10	10,10	10,10	10,10
YRPDV	8.5,13.5	8.5,18.5	8.0,11.0	8.0,11.0
MPEQYR	9	9,15	12	12,17
NENV	8	8	8	8

E. Running PRICE2

Operating PRICE2 via the Univesity of Alberta's Amdahl 470v/7 computer is a simple two command sequence. These are:

```
$run *fortg scards=rdmo:price2 spunch=-file t=4s
$run -file 5=*source* 6=*sink* 7=-output 6=4s
```

If these two commands are properly entered, PRICE2 will begin running by briefly explaining how information is

entered into the program. "Free-format" data entry applies, which requires the least degree of regulation of how information is given to the computer. These four rules apply:

1. Character strings are enclosed by parentheses.
2. A blank space or comma separates input values.
3. Integers cannot contain decimal points, while real numbers must contain a decimal point.
4. Each data entry request must be accomplised on a single line.

To further clarify how data are entered into PRICE2 an example accompanies each data request made by the program. Once all necessary data have been given to PRICE2, the program will make the relevant calculations and store the results in the output file named "-output" when the previously stated commands are given.

F. Sample data output from PRICE2

```
OPEN PIT MINE AT       Jan. 1990
  MARSHALL & SWIFT INDEX (MINING & MILLING)     1100.0
  WHOLESALE PRICE INDEX (INDUSTRIALS)           1220.0
INPUT COSTS VALID AT  Oct. 1979
  MARSHALL & SWIFT INDEX (MINING & MILLING)      613.4
  WHOLESALE PRICE INDEX (INDUSTRIALS)            687.0
TOTAL ORE RESERVES (MILLION S.TON)                1.825
AVERAGE ORE GRADE (URANIUM OXIDE %)               0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY)     500.
MINE OPERATING LIFE (YEARS)                      10
YEARS LEAD TIME, YEAR 0 TO START-UP              10

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)  10, 10
  PRICE ESCALATION RATES (PERCENT)                3.00, 5.00
  COST  ESCALATION RATES (PERCENT)                5.00, 5.00
OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS
AGAINST.
DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO
STRAIGHT LINE.
```

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	36.19
10.0	18.00	38.90
10.0	21.00	42.01
10.0	24.00	45.56
10.0	27.00	49.61
20.0	15.00	42.89
20.0	18.00	45.95
20.0	21.00	49.44
20.0	24.00	53.44
20.0	27.00	58.01
30.0	15.00	49.63
30.0	18.00	53.04
30.0	21.00	56.93
30.0	24.00	61.39
30.0	27.00	66.51
40.0	15.00	56.39
40.0	18.00	60.16
40.0	21.00	64.48
40.0	24.00	69.41
40.0	27.00	75.07
50.0	15.00	63.18
50.0	18.00	67.33

50.0	21.00	72.06
50.0	24.00	77.45
50.0	27.00	83.64
60.0	15.00	70.01
60.0	18.00	74.50
60.0	21.00	79.64
60.0	24.00	85.50
60.0	27.00	92.22
70.0	15.00	76.84
70.0	18.00	81.67
70.0	21.00	87.23
70.0	24.00	93.55
70.0	27.00	100.81
80.0	15.00	83.66
80.0	18.00	88.85
80.0	21.00	94.81
80.0	24.00	101.61
80.0	27.00	109.40

OPEN PIT MINE AT Jan. 1990	
MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0
INPUT COSTS VALID AT Oct. 1979	
MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	3.650
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	1000.
MINE OPERATING LIFE (YEARS)	10
YEARS LEAD TIME, YEAR 0 TO START-UP	10
FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	31.42
10.0	18.00	33.83
10.0	21.00	36.61
10.0	24.00	39.82
10.0	27.00	43.51
20.0	15.00	38.14
20.0	18.00	40.90
20.0	21.00	44.07
20.0	24.00	47.73
20.0	27.00	51.95
30.0	15.00	44.89
30.0	18.00	48.01
30.0	21.00	51.59
30.0	24.00	55.72
30.0	27.00	60.48
40.0	15.00	51.67
40.0	18.00	55.16
40.0	21.00	59.16
40.0	24.00	63.75
40.0	27.00	69.04
50.0	15.00	58.49
50.0	18.00	62.33
50.0	21.00	66.74
50.0	24.00	71.79
50.0	27.00	77.61

60.0	15.00	65.32
60.0	18.00	69.50
60.0	21.00	74.33
60.0	24.00	79.84
60.0	27.00	86.20
70.0	15.00	72.15
70.0	18.00	76.68
70.0	21.00	81.91
70.0	24.00	87.90
70.0	27.00	94.79
80.0	15.00	78.98
80.0	18.00	83.86
80.0	21.00	89.49
80.0	24.00	95.95
80.0	27.00	103.38

OPEN PIT MINE AT	Jan. 1990	
MARSHALL & SWIFT INDEX (MINING & MILLING)		1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)		1220.0
INPUT COSTS VALID AT	Oct. 1979	
MARSHALL & SWIFT INDEX (MINING & MILLING)		613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)		687.0
TOTAL ORE RESERVES (MILLION S.TON)		7.300
AVERAGE ORE GRADE (URANIUM OXIDE %)		0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY)		2000.
MINE OPERATING LIFE (YEARS)		10
YEARS LEAD TIME, YEAR 0 TO START-UP		10

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE \$/LB U. OX.
10.0	15.00	27.48
10.0	18.00	29.63
10.0	21.00	32.12
10.0	24.00	35.02
10.0	27.00	38.38
20.0	15.00	33.90
20.0	18.00	36.39
20.0	21.00	39.27
20.0	24.00	42.61
20.0	27.00	46.51
30.0	15.00	40.35
30.0	18.00	43.19
30.0	21.00	46.47
30.0	24.00	50.28
30.0	27.00	54.69
40.0	15.00	46.83
40.0	18.00	50.02
40.0	21.00	53.71
40.0	24.00	57.96
40.0	27.00	62.90
50.0	15.00	53.34
50.0	18.00	56.86
50.0	21.00	60.95
50.0	24.00	65.65
50.0	27.00	71.11

60.0	15.00	59.84
60.0	18.00	63.70
60.0	21.00	68.19
60.0	24.00	73.35
60.0	27.00	79.33
70.0	15.00	66.35
70.0	18.00	70.55
70.0	21.00	75.43
70.0	24.00	81.05
70.0	27.00	87.56
80.0	15.00	72.86
80.0	18.00	77.39
80.0	21.00	82.66
80.0	24.00	88.75
80.0	27.00	95.78

UNDERGROUND MINE AT Jan. 1990
MARSHALL & SWIFT INDEX (MINING & MILLING) 1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS) 1220.0

INPUT COSTS VALID AT Oct. 1979
MARSHALL & SWIFT INDEX (MINING & MILLING) 613.4
WHOLESALE PRICE INDEX (INDUSTRIALS) 687.0

TOTAL ORE RESERVES (MILLION S.TON) 2.433
AVERAGE ORE GRADE (URANIUM OXIDE %) 0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY) 500.
MINE OPERATING LIFE (YEARS) 10
YEARS LEAD TIME, YEAR 0 TO START-UP 11

FIRST PERIOD PRICE AND COST ESCALATION (YEARS) 10, 10
PRICE ESCALATION RATES (PERCENT) 3.00, 5.00
COST ESCALATION RATES (PERCENT) 5.00, 5.00
OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	44.38
10.0	18.00	47.46
10.0	21.00	51.03
10.0	24.00	55.16
10.0	27.00	59.97
20.0	15.00	46.64
20.0	18.00	50.13
20.0	21.00	54.19
20.0	24.00	58.91
20.0	27.00	64.42
30.0	15.00	48.92
30.0	18.00	52.84
30.0	21.00	57.40
30.0	24.00	62.71
30.0	27.00	68.92
40.0	15.00	51.21
40.0	18.00	55.57
40.0	21.00	60.63
40.0	24.00	66.54
40.0	27.00	73.47
50.0	15.00	53.51
50.0	18.00	58.31
50.0	21.00	63.88
50.0	24.00	70.39

50.0	27.00	78.02
60.0	15.00	55.82
60.0	18.00	61.05
60.0	21.00	67.23
60.0	24.00	74.24
60.0	27.00	82.59
70.0	15.00	58.13
70.0	18.00	63.79
70.0	21.00	70.38
70.0	24.00	78.09
70.0	27.00	87.15
80.0	15.00	60.43
80.0	18.00	66.52
80.0	21.00	73.63
80.0	24.00	81.95
80.0	27.00	91.72
90.0	15.00	62.74
90.0	18.00	69.26
90.0	21.00	76.88
90.0	24.00	85.80
90.0	27.00	96.29
100.0	15.00	65.05
100.0	18.00	72.00
100.0	21.00	80.14
100.0	24.00	89.66
100.0	27.00	100.86
120.0	15.00	69.67
120.0	18.00	77.48
120.0	21.00	86.65
120.0	24.00	97.37
120.0	27.00	110.21
140.0	15.00	74.29
140.0	18.00	82.96
140.0	21.00	93.16
140.0	24.00	105.10
140.0	27.00	119.87

UNDERGROUND MINE AT Jan. 1990	
MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0
INPUT COSTS VALID AT Oct. 1979	
MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	4.867
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	1000.
MINE OPERATING LIFE (YEARS)	10
YEARS LEAD TIME, YEAR 0 TO START-UP	11
FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	38.50
10.0	18.00	41.27
10.0	21.00	44.50
10.0	24.00	48.28
10.0	27.00	52.72
20.0	15.00	40.73
20.0	18.00	43.92
20.0	21.00	47.65
20.0	24.00	52.02
20.0	27.00	57.16
30.0	15.00	42.99
30.0	18.00	46.61
30.0	21.00	50.84
30.0	24.00	55.80
30.0	27.00	61.65
40.0	15.00	45.25
40.0	18.00	49.31
40.0	21.00	54.05
40.0	24.00	59.61
40.0	27.00	66.16
50.0	15.00	47.53
50.0	18.00	52.02
50.0	21.00	57.26
50.0	24.00	63.42

50.0	27.00	70.69
60.0	15.00	49.80
60.0	18.00	54.73
60.0	21.00	60.48
60.0	24.00	67.25
60.0	27.00	75.23
70.0	15.00	52.08
70.0	18.00	57.43
70.0	21.00	63.70
70.0	24.00	71.07
70.0	27.00	79.76
80.0	15.00	54.36
80.0	18.00	60.14
80.0	21.00	66.92
80.0	24.00	74.89
80.0	27.00	84.30
90.0	15.00	56.63
90.0	18.00	62.85
90.0	21.00	70.15
90.0	24.00	78.72
90.0	27.00	88.88
100.0	15.00	58.91
100.0	18.00	65.55
100.0	21.00	73.37
100.0	24.00	82.54
100.0	27.00	93.57
120.0	15.00	63.47
120.0	18.00	70.97
120.0	21.00	79.82
120.0	24.00	90.22
120.0	27.00	103.19
140.0	15.00	68.03
140.0	18.00	76.39
140.0	21.00	86.26
140.0	24.00	98.14
140.0	27.00	113.04

UNDERGROUND MINE AT Jan. 1990

MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0

INPUT COSTS VALID AT Oct. 1979

MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	9.733
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	2000.
MINE OPERATING LIFE (YEARS)	10
YEARS LEAD TIME, YEAR 0 TO START-UP	11

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	33.18
10.0	18.00	35.65
10.0	21.00	38.55
10.0	24.00	41.97
10.0	27.00	46.04
20.0	15.00	35.22
20.0	18.00	38.08
20.0	21.00	41.44
20.0	24.00	45.43
20.0	27.00	50.16
30.0	15.00	38.14
30.0	18.00	41.10
30.0	21.00	44.36
30.0	24.00	48.91
30.0	27.00	54.31
40.0	15.00	39.32
40.0	18.00	42.99
40.0	21.00	47.30
40.0	24.00	52.41
40.0	27.00	58.48
50.0	15.00	41.38
50.0	18.00	45.44
50.0	21.00	50.24
50.0	24.00	55.91

50.0	27.00	62.66
60.0	15.00	43.44
60.0	18.00	47.90
60.0	21.00	53.18
60.0	24.00	59.42
60.0	27.00	66.84
70.0	15.00	45.50
70.0	18.00	50.36
70.0	21.00	56.12
70.0	24.00	62.93
70.0	27.00	71.03
80.0	15.00	47.56
80.0	18.00	52.82
80.0	21.00	59.06
80.0	24.00	66.44
80.0	27.00	75.23
90.0	15.00	49.62
90.0	18.00	55.28
90.0	21.00	62.01
90.0	24.00	69.95
90.0	27.00	79.54
100.0	15.00	51.68
100.0	18.00	57.74
100.0	21.00	64.95
100.0	24.00	73.46
100.0	27.00	83.95
120.0	15.00	55.81
120.0	18.00	62.66
120.0	21.00	70.83
120.0	24.00	80.59
120.0	27.00	92.96
140.0	15.00	59.93
140.0	18.00	67.58
140.0	21.00	76.71
140.0	24.00	88.01
140.0	27.00	102.16

OPEN PIT MINE AT Jan. 1990

MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0

INPUT COSTS VALID AT Oct. 1979

MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	1.825
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.050
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	1000.
MINE OPERATING LIFE (YEARS)	5
YEARS LEAD TIME, YEAR 0 TO START-UP	10

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	54.97
10.0	18.00	57.07
10.0	21.00	59.36
10.0	24.00	61.86
10.0	27.00	64.59
20.0	15.00	67.35
20.0	18.00	69.76
20.0	21.00	72.39
20.0	24.00	75.26
20.0	27.00	78.39
30.0	15.00	79.84
30.0	18.00	82.58
30.0	21.00	85.57
30.0	24.00	88.82
30.0	27.00	92.37
40.0	15.00	92.39
40.0	18.00	95.46
40.0	21.00	98.84
40.0	24.00	102.55
40.0	27.00	106.59
50.0	15.00	105.12

OPEN PIT MINE AT Jan. 1990
MARSHALL & SWIFT INDEX (MINING & MILLING) 1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS) 1220.0

INPUT COSTS VALID AT Oct. 1979
MARSHALL & SWIFT INDEX (MINING & MILLING) 613.4
WHOLESALE PRICE INDEX (INDUSTRIALS) 687.0
TOTAL ORE RESERVES (MILLION S.TON) 3.650
AVERAGE ORE GRADE (URANIUM OXIDE %) 0.050
ORE PRODUCTION RATE (TON PER CALENDAR DAY) 2000.
MINE OPERATING LIFE (YEARS) 5
YEARS LEAD TIME, YEAR 0 TO START-UP 10

FIRST PERIOD PRICE AND COST ESCALATION (YEARS) 10, 10
PRICE ESCALATION RATES (PERCENT) 3.00, 5.00
COST ESCALATION RATES (PERCENT) 5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	46.92
10.0	18.00	48.71
10.0	21.00	50.67
10.0	24.00	52.82
10.0	27.00	55.18
20.0	15.00	58.41
20.0	18.00	60.48
20.0	21.00	62.75
20.0	24.00	65.24
20.0	27.00	67.98
30.0	15.00	69.97
30.0	18.00	72.33
30.0	21.00	74.93
30.0	24.00	77.78
30.0	27.00	80.91
40.0	15.00	81.58
40.0	18.00	84.29
40.0	21.00	87.26
40.0	24.00	90.51
40.0	27.00	94.09
50.0	15.00	93.42
50.0	18.00	96.42
50.0	21.00	99.72

50.0	24.00	103.34
50.0	27.00	107.30
60.0	15.00	105.27

OPEN PIT MINE AT Jan. 1990

MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0

INPUT COSTS VALID AT Oct. 1979

MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	5.475
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.050
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	3000.
MINE OPERATING LIFE (YEARS)	5
YEARS LEAD TIME, YEAR 0 TO START-UP	10

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	42.95
10.0	18.00	44.57
10.0	21.00	46.36
10.0	24.00	48.33
10.0	27.00	50.50
20.0	15.00	54.02
20.0	18.00	55.91
20.0	21.00	58.00
20.0	24.00	60.30
20.0	27.00	62.85
30.0	15.00	65.15
30.0	18.00	67.32
30.0	21.00	69.72
30.0	24.00	72.36
30.0	27.00	75.31
40.0	15.00	76.36
40.0	18.00	78.86
40.0	21.00	81.61
40.0	24.00	84.64
40.0	27.00	87.98
50.0	15.00	87.75
50.0	18.00	90.52
50.0	21.00	93.58

50.0	24.00	96.96
50.0	27.00	100.68
60.0	15.00	99.14
60.0	18.00	102.19
60.0	21.00	105.57
60.0	24.00	109.29
60.0	27.00	113.39
70.0	15.00	110.54

OPEN PIT MINE AT Jan. 1990
MARSHALL & SWIFT INDEX (MINING & MILLING) 1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS) 1220.0

INPUT COSTS VALID AT Oct. 1979
MARSHALL & SWIFT INDEX (MINING & MILLING) 613.4
WHOLESALE PRICE INDEX (INDUSTRIALS) 687.0
TOTAL ORE RESERVES (MILLION S.TON) 0.912
AVERAGE ORE GRADE (URANIUM OXIDE %) 0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY) 500.
MINE OPERATING LIFE (YEARS) 5
YEARS LEAD TIME, YEAR 0 TO START-UP 10

FIRST PERIOD PRICE AND COST ESCALATION (YEARS) 10, 10
PRICE ESCALATION RATES (PERCENT) 3.00, 5.00
COST ESCALATION RATES (PERCENT) 5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	34.96
10.0	18.00	36.63
10.0	21.00	38.49
10.0	24.00	40.60
10.0	27.00	42.97
20.0	15.00	41.84
20.0	18.00	43.75
20.0	21.00	45.90
20.0	24.00	48.32
20.0	27.00	51.05
30.0	15.00	48.79
30.0	18.00	50.96
30.0	21.00	53.40
30.0	24.00	56.14
30.0	27.00	59.24
40.0	15.00	55.77
40.0	18.00	58.21
40.0	21.00	60.94
40.0	24.00	64.05
40.0	27.00	67.57
50.0	15.00	62.79
50.0	18.00	65.54
50.0	21.00	68.62

50.0	24.00	72.07
50.0	27.00	75.97
60.0	15.00	69.94
60.0	18.00	72.94
60.0	21.00	76.32
60.0	24.00	80.11
60.0	27.00	84.38
70.0	15.00	77.08
70.0	18.00	80.36
70.0	21.00	84.03
70.0	24.00	88.15
70.0	27.00	92.80
80.0	15.00	84.23
80.0	18.00	87.77
80.0	21.00	91.74
80.0	24.00	96.20
80.0	27.00	101.23

OPEN PIT MINE AT Jan. 1990

MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0

INPUT COSTS VALID AT Oct. 1979

MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	1.825
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	1000.
MINE OPERATING LIFE (YEARS)	5
YEARS LEAD TIME, YEAR 0 TO START-UP	10

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	30.45
10.0	18.00	31.94
10.0	21.00	33.63
10.0	24.00	35.55
10.0	27.00	37.73
20.0	15.00	37.36
20.0	18.00	39.10
20.0	21.00	41.08
20.0	24.00	43.32
20.0	27.00	45.86
30.0	15.00	44.32
30.0	18.00	46.33
30.0	21.00	48.59
30.0	24.00	51.16
30.0	27.00	54.11
40.0	15.00	51.31
40.0	18.00	53.62
40.0	21.00	56.22
40.0	24.00	59.17
40.0	27.00	62.50
50.0	15.00	58.45
50.0	18.00	61.02
50.0	21.00	63.92
50.0	24.00	67.20

50.0	27.00	70.90
60.0	15.00	65.59
60.0	18.00	68.43
60.0	21.00	71.63
60.0	24.00	75.24
60.0	27.00	79.32
70.0	15.00	72.74
70.0	18.00	75.84
70.0	21.00	79.34
70.0	24.00	83.28
70.0	27.00	87.74
80.0	15.00	79.89
80.0	18.00	83.26
80.0	21.00	87.05
80.0	24.00	91.33
80.0	27.00	96.16

OPEN PIT MINE AT Jan. 1990

MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0

INPUT COSTS VALID AT Oct. 1979

MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	3.650
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	2000.
MINE OPERATING LIFE (YEARS)	5
YEARS LEAD TIME, YEAR 0 TO START-UP	10

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.
DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	26.72
10.0	18.00	28.06
10.0	21.00	29.59
10.0	24.00	31.34
10.0	27.00	33.35
20.0	15.00	33.36
20.0	18.00	34.94
20.0	21.00	36.75
20.0	24.00	38.81
20.0	27.00	41.17
30.0	15.00	40.03
30.0	18.00	41.86
30.0	21.00	43.95
30.0	24.00	46.37
30.0	27.00	49.13
40.0	15.00	46.77
40.0	18.00	48.89
40.0	21.00	51.30
40.0	24.00	54.05
40.0	27.00	57.17
50.0	15.00	53.59
50.0	18.00	55.97
50.0	21.00	58.67
50.0	24.00	61.73
50.0	27.00	65.22

60.0	15.00	60.42
60.0	18.00	63.05
60.0	21.00	66.03
60.0	24.00	69.42
60.0	27.00	73.27
70.0	15.00	67.26
70.0	18.00	70.14
70.0	21.00	73.41
70.0	24.00	77.11
70.0	27.00	81.33
80.0	15.00	74.09
80.0	18.00	77.23
80.0	21.00	80.78
80.0	24.00	84.81
80.0	27.00	89.39

UNDERGROUND MINE AT Jan. 1990

MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0

INPUT COSTS VALID AT Oct. 1979

MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	2.433
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.050
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	1000.
MINE OPERATING LIFE (YEARS)	5
YEARS LEAD TIME, YEAR 0 TO START-UP	11

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	60.86
10.0	18.00	63.13
10.0	21.00	65.60
10.0	24.00	68.31
10.0	27.00	71.28
20.0	15.00	62.89
20.0	18.00	65.38
20.0	21.00	68.10
20.0	24.00	71.08
20.0	27.00	74.37
30.0	15.00	65.01
30.0	18.00	67.72
30.0	21.00	70.71
30.0	24.00	73.99
30.0	27.00	77.62
40.0	15.00	67.16
40.0	18.00	70.12
40.0	21.00	73.37
40.0	24.00	76.96
40.0	27.00	80.94
50.0	15.00	69.34
50.0	18.00	72.54
50.0	21.00	76.06
50.0	24.00	79.96
50.0	27.00	84.29

60.0	15.00	71.52
60.0	18.00	74.97
60.0	21.00	78.76
60.0	24.00	82.97
60.0	27.00	87.66
70.0	15.00	73.71
70.0	18.00	77.40
70.0	21.00	81.47
70.0	24.00	86.00
70.0	27.00	91.04
80.0	15.00	75.91
80.0	18.00	79.84
80.0	21.00	84.19
80.0	24.00	89.02
80.0	27.00	94.42
90.0	15.00	78.10
90.0	18.00	82.28
90.0	21.00	86.90
90.0	24.00	92.05
90.0	27.00	97.80
100.0	15.00	80.30
100.0	18.00	84.72
100.0	21.00	89.62
100.0	24.00	95.08
100.0	27.00	101.19
120.0	15.00	84.69
120.0	18.00	89.59
120.0	21.00	95.05
120.0	24.00	101.14
120.0	27.00	107.96
140.0	15.00	89.08
140.0	18.00	94.47
140.0	21.00	100.48
140.0	24.00	107.20
140.0	27.00	114.74

UNDERGROUND MINE AT Jan. 1990

MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0

INPUT COSTS VALID AT Oct. 1979
STRAIGHT LINE.

MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	4.867
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.050
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	2000.
MINE OPERATING LIFE (YEARS)	5
YEARS LEAD TIME, YEAR 0 TO START-UP	11

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	51.73
10.0	18.00	53.68
10.0	21.00	55.81
10.0	24.00	58.16
10.0	27.00	60.76
20.0	15.00	53.61
20.0	18.00	55.77
20.0	21.00	58.14
20.0	24.00	60.76
20.0	27.00	63.67
30.0	15.00	55.55
30.0	18.00	57.92
30.0	21.00	60.54
30.0	24.00	63.45
30.0	27.00	66.69
40.0	15.00	57.52
40.0	18.00	60.11
40.0	21.00	62.99
40.0	24.00	66.18
40.0	27.00	69.75
50.0	15.00	59.49
50.0	18.00	62.32
50.0	21.00	65.45
50.0	24.00	68.93

50.0	27.00	72.83
60.0	15.00	61.48
60.0	18.00	64.53
60.0	21.00	67.91
60.0	24.00	71.69
60.0	27.00	75.93
70.0	15.00	63.47
70.0	18.00	66.75
70.0	21.00	70.39
70.0	24.00	74.46
70.0	27.00	79.03
80.0	15.00	65.46
80.0	18.00	68.96
80.0	21.00	72.86
80.0	24.00	77.23
80.0	27.00	82.13
90.0	15.00	67.45
90.0	18.00	71.18
90.0	21.00	75.34
90.0	24.00	80.00
90.0	27.00	85.24
100.0	15.00	69.45
100.0	18.00	73.40
100.0	21.00	77.82
100.0	24.00	82.77
100.0	27.00	88.34
120.0	15.00	73.43
120.0	18.00	77.84
120.0	21.00	82.77
120.0	24.00	88.31
120.0	27.00	94.56
140.0	15.00	77.42
140.0	18.00	82.28
140.0	21.00	87.73
140.0	24.00	93.86
140.0	27.00	100.77

UNDERGROUND MINE AT Jan. 1990

MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0

INPUT COSTS VALID AT Oct. 1979

MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	7.300
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.050
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	3000.
MINE OPERATING LIFE (YEARS)	5
YEARS LEAD TIME, YEAR 0 TO START-UP	11

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	46.75
10.0	18.00	48.52
10.0	21.00	50.48
10.0	24.00	52.64
10.0	27.00	55.04
20.0	15.00	48.51
20.0	18.00	50.48
20.0	21.00	52.66
20.0	24.00	55.08
20.0	27.00	57.79
30.0	15.00	50.31
30.0	18.00	52.49
30.0	21.00	54.91
30.0	24.00	57.60
30.0	27.00	60.61
40.0	15.00	52.14
40.0	18.00	54.53
40.0	21.00	57.18
40.0	24.00	60.15
40.0	27.00	63.48
50.0	15.00	53.97
50.0	18.00	56.57
50.0	21.00	59.47
50.0	24.00	62.71

50.0	27.00	66.36
60.0	15.00	55.81
60.0	18.00	58.62
60.0	21.00	61.76
60.0	24.00	65.28
60.0	27.00	69.25
70.0	15.00	57.65
70.0	18.00	60.68
70.0	21.00	64.06
70.0	24.00	67.86
70.0	27.00	72.14
80.0	15.00	59.49
80.0	18.00	62.73
80.0	21.00	66.36
80.0	24.00	70.44
80.0	27.00	75.04
90.0	15.00	61.33
90.0	18.00	64.79
90.0	21.00	68.66
90.0	24.00	73.02
90.0	27.00	77.94
100.0	15.00	63.18
100.0	18.00	66.85
100.0	21.00	70.96
100.0	24.00	75.60
100.0	27.00	80.84
120.0	15.00	66.87
120.0	18.00	70.96
120.0	21.00	75.56
120.0	24.00	80.75
120.0	27.00	86.64
140.0	15.00	70.55
140.0	18.00	75.08
140.0	21.00	80.16
140.0	24.00	85.91
140.0	27.00	92.44

UNDERGROUND MINE AT Jan. 1990

MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0

INPUT COSTS VALID AT Oct. 1979

MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	1.217
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	500.
MINE OPERATING LIFE (YEARS)	5
YEARS LEAD TIME, YEAR 0 TO START-UP	11

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	42.60
10.0	18.00	44.51
10.0	21.00	46.67
10.0	24.00	49.14
10.0	27.00	51.96
20.0	15.00	44.46
20.0	18.00	46.60
20.0	21.00	49.04
20.0	24.00	51.83
20.0	27.00	55.05
30.0	15.00	46.36
30.0	18.00	48.75
30.0	21.00	51.48
30.0	24.00	54.61
30.0	27.00	58.22
40.0	15.00	48.29
40.0	18.00	50.93
40.0	21.00	53.96
40.0	24.00	57.43
40.0	27.00	61.44
50.0	15.00	50.23
50.0	18.00	53.13
50.0	21.00	56.45
50.0	24.00	60.27
50.0	27.00	64.68

60.0	15.00	52.17
60.0	18.00	55.33
60.0	21.00	58.94
60.0	24.00	63.11
60.0	27.00	67.93
70.0	15.00	54.12
70.0	18.00	57.53
70.0	21.00	61.45
70.0	24.00	65.96
70.0	27.00	71.19
80.0	15.00	56.07
80.0	18.00	59.74
80.0	21.00	63.95
80.0	24.00	68.81
80.0	27.00	74.45
90.0	15.00	58.02
90.0	18.00	61.95
90.0	21.00	66.46
90.0	24.00	71.67
90.0	27.00	77.71
100.0	15.00	59.98
100.0	18.00	64.15
100.0	21.00	68.96
100.0	24.00	74.52
100.0	27.00	80.97
120.0	15.00	63.88
120.0	18.00	68.57
120.0	21.00	73.97
120.0	24.00	80.23
120.0	27.00	87.50
140.0	15.00	67.79
140.0	18.00	72.99
140.0	21.00	78.99
140.0	24.00	85.94
140.0	27.00	94.02

UNDERGROUND MINE AT Jan. 1990

MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0

INPUT COSTS VALID AT Oct. 1979

MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	2.433
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	1000.
MINE OPERATING LIFE (YEARS)	5
YEARS LEAD TIME, YEAR 0 TO START-UP	11

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	36.94
10.0	18.00	38.67
10.0	21.00	40.65
10.0	24.00	42.92
10.0	27.00	45.55
20.0	15.00	38.78
20.0	18.00	40.75
20.0	21.00	43.02
20.0	24.00	45.62
20.0	27.00	48.64
30.0	15.00	40.66
30.0	18.00	42.89
30.0	21.00	45.44
30.0	24.00	48.38
30.0	27.00	51.80
40.0	15.00	42.56
40.0	18.00	45.04
40.0	21.00	47.89
40.0	24.00	51.18
40.0	27.00	55.00
50.0	15.00	44.47
50.0	18.00	47.21
50.0	21.00	50.35
50.0	24.00	53.99

50.0	27.00	58.21
60.0	15.00	46.39
60.0	18.00	49.38
60.0	21.00	52.82
60.0	24.00	56.80
60.0	27.00	61.43
70.0	15.00	48.31
70.0	18.00	51.55
70.0	21.00	55.29
70.0	24.00	59.62
70.0	27.00	64.66
80.0	15.00	50.23
80.0	18.00	53.73
80.0	21.00	57.76
80.0	24.00	62.44
80.0	27.00	67.89
90.0	15.00	52.15
90.0	18.00	55.90
90.0	21.00	60.23
90.0	24.00	65.26
90.0	27.00	71.11
100.0	15.00	54.07
100.0	18.00	58.08
100.0	21.00	62.71
100.0	24.00	68.08
100.0	27.00	74.34
120.0	15.00	57.91
120.0	18.00	62.43
120.0	21.00	67.65
120.0	24.00	73.72
120.0	27.00	80.80
140.0	15.00	61.75
140.0	18.00	66.78
140.0	21.00	72.60
140.0	24.00	79.37
140.0	27.00	87.26

UNDERGROUND MINE AT Jan. 1990

MARSHALL & SWIFT INDEX (MINING & MILLING)	1100.0
WHOLESALE PRICE INDEX (INDUSTRIALS)	1220.0

INPUT COSTS VALID AT Oct. 1979

MARSHALL & SWIFT INDEX (MINING & MILLING)	613.4
WHOLESALE PRICE INDEX (INDUSTRIALS)	687.0
TOTAL ORE RESERVES (MILLION S.TON)	4.867
AVERAGE ORE GRADE (URANIUM OXIDE %)	0.100
ORE PRODUCTION RATE (TON PER CALENDAR DAY)	2000.
MINE OPERATING LIFE (YEARS)	5
YEARS LEAD TIME, YEAR 0 TO START-UP	11

FIRST PERIOD PRICE AND COST ESCALATION (YEARS)	10, 10
PRICE ESCALATION RATES (PERCENT)	3.00, 5.00
COST ESCALATION RATES (PERCENT)	5.00, 5.00

OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE TAX BENEFITS AGAINST.

DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT LINE.

DEPTH TO THICKNESS RATIO	RATE OF RETURN (PERCENT)	REQUIRED PRICE $/LB U. OX.
10.0	15.00	31.85
10.0	18.00	33.40
10.0	21.00	35.18
10.0	24.00	37.26
10.0	27.00	39.68
20.0	15.00	33.53
20.0	18.00	35.31
20.0	21.00	37.37
20.0	24.00	39.76
20.0	27.00	42.56
30.0	15.00	35.25
30.0	18.00	37.26
30.0	21.00	39.60
30.0	24.00	42.31
30.0	27.00	45.49
40.0	15.00	36.97
40.0	18.00	39.23
40.0	21.00	41.84
40.0	24.00	44.88
40.0	27.00	48.44
50.0	15.00	38.71
50.0	18.00	41.20
50.0	21.00	44.09
50.0	24.00	47.46

50.0	27.00	51.41
60.0	15.00	40.44
60.0	18.00	43.17
60.0	21.00	46.34
60.0	24.00	50.04
60.0	27.00	54.38
70.0	15.00	42.18
70.0	18.00	45.15
70.0	21.00	48.60
70.0	24.00	52.63
70.0	27.00	57.35
80.0	15.00	43.92
80.0	18.00	47.13
80.0	21.00	50.86
80.0	24.00	55.21
80.0	27.00	60.33
90.0	15.00	45.65
90.0	18.00	49.10
90.0	21.00	53.11
90.0	24.00	57.80
90.0	27.00	63.30
100.0	15.00	47.39
100.0	18.00	51.08
100.0	21.00	55.37
100.0	24.00	60.39
100.0	27.00	66.28
120.0	15.00	50.87
120.0	18.00	55.04
120.0	21.00	59.89
120.0	24.00	65.57
120.0	27.00	72.23
140.0	15.00	54.34
140.0	18.00	59.00
140.0	21.00	64.41
140.0	24.00	70.74
140.0	27.00	78.20

A. The PRICE2 Program

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C   PRICE.FOR
C   ---------
C
C   WRITTEN BY TREVOR R. ELLIS, DEPARTMENT OF MINERAL
C   ECONOMICS, COLORADO SCHOOL OF MINES, JANUARY 1978.
C
C   THE PROGRAM WAS MODIFIED AT THE UNIVERSITY OF ALBERTA
C   BY CHRIS VAN DYKE.
C
C   THIS PROGRAM GENERATES THE MINIMUM DOLLAR PER POUND PRICE OF
C   URANIUM OXIDE (YELLOWCAKE) NECESSARY FOR ECONOMICAL CONTINUATION
C   OF EXPLORATION AND DEVELOPMENT WORK TOWARDS CONVENTIONAL MINING OF
C   SANDSTONE-TYPE URANIUM DEPOSIT.
C
C   THE PROGRAM CONTAINS SIX SUBROUTINES. THESE ARE OPEN, UNDER,
C   DEDUCT, INVEST, NETPV AND INCOME.
C
C
C
C   MAIN PROGRAM
C
C
C   INITIALLY THE PROGRAM INTERACTIVELY REQUESTS GENERALIZED
C   DATA NEEDED FOR EITHER UNDERGROUND OR OPEN PIT MINING.
C
C   THE INPUT DATA IS ADJUSTED FOR USE IN THE SUBROUTINES. THEN THE
C   SUBROUTINE 'UNDER' IS SELECTED FOR UNDERGROUND MINING OR 'OPEN' FOR
C   OPEN PIT MINING.
C
C   THE MAIN PROGRAM IS RETURNED TO AT THE END OF THE RUN TO WRITE OUT
C   THE RESULTS AND FOR AUTOMATIC RERUNNING IF MORE FILES ARE LISTED.
C
C
      INTEGER NENV
      DIMENSION ANCOST(11), AVEGRD(6), COSTPT(10), ESC(4), ESCALN(4)
      DIMENSION PARAM(12), ROR(6), ROR1(6), DTREQ(40)
      INTEGER FLAG(7)
      DIMENSION NYEAR(10,2), PRICE(5,40)
      LOGICAL*1 DATEC(10), DATEP(10)
C
      REAL MILREC(6)
C
C
C
      WRITE(6,9)
9     FORMAT('THIS PROGRAM IS INTERACTIVE, DATA INPUT IS',/,
```

```
     &'DDNE DN THE TERMINAL.  THE "FREE FDRMAT SYSTEM"',/,
     &'ALLDWS A LARGE DEGREE DF FREEDDM IN FORMATTING.',/,
     &'THESE SIMPLE RULES APPLY',/,
     &'  1. CHARACTER STRINGS SHOULD BE ENCLDSED WITH DDUBLE ',/,
     &'QUDTE PUNCTUATION MARKS',/,
     &'  2. DATA VALUES ARE SEPARATED BY BLANKS DR CDMMAS',/,
     &'  3. EACH DATA REQUEST MUST BE ACCOMPLISHED DN A SINGLE LINE',/,
     &'  4. REAL VALUES MUST INCLUDE A DECIMAL PDINT, AND INTEGERS DD',/,
     &'INCLUDE A DECIMAL POINT.',/,
     &/,'AN EXAMPLE IS PROVIDED WITH EACH REQUEST FOR INFDRMATIDN')
C
C
C  DATEC = DATE CDST INPUTS TAKEN; DATEP = DATE PRICE DUTPUT VALID
C  FDR.
C
      WRITE(6,11)
11    FORMAT('ENTER DATEC=DATE COST INPUTS TAKEN, E.G. "JAN 1974"')
      CALL FREAD(5, 'STRING:', DATEC, 10)
      WRITE(6,12)
12    FDRMAT('ENTER DATEP=DATE PRICE OUTPUT VALID FDR, ',/,
     &'EXAMPLE: "SEPT 1977"')
      CALL FREAD(5, 'STRING:', DATEP, 10)
C
C     INPUT 12 GENERAL PARAMETERS:
C
      WRITE(6,13)
13    FDRMAT('ENTER THE 12 GENERAL PARAMETERS PARAM(12). ',/,
     &'SEE THE USERS MANUAL FDR A DESCRIPTIDN',/,
     &'DF THE INFORMATION NEEDED.  AN EXAMPLE IS GIVEN'/,
     &'348. 526.6 132.2 197.8 12. 22. 50. 10. 70. 56. 100. 50.',/,
     &'ALL VALUES ARE REAL, I.E. MUST BE ACCDMPANIED BY A ',/,
     &'DECIMAL POINT')
      CALL FREAD(5, 'R V:', PARAM, 12)
C
C     DIVIDE BOTH PERCENTAGES AND CENTS BY 100:
C
      DD 50 I = 5,9
      PARAM(I) = PARAM(I) / 100.
50    CONTINUE
      PARAM(12) = PARAM(12) / 100.
C
C     INPUT MILL DEPRECIATIDN LIFE (YEARS):
C
      WRITE(6,14)
14    FDRMAT('ENTER MILDEP=MILL DEPRECIATIDN LIFE IN YEARS.',/,
     &'AN INTEGER IS REQUIRED.  EXAMPLE:  20')
      CALL FREAD(5, 'I:',MILDEP)
C
```

```
C     INPUT FLAG(1). POSITIVE FOR STRAIGHT LINE OEPRECIATION; NEGATIVE
C  FOR OOUBLE OECLINING BALANCE WITH SWITCHING TO STRAIGHT LINE WHEN
C  BENEFICIAL.
C
      WRITE(6,15)
15    FORMAT('ENTER FLAG(1).  A POSITIVE INTEGER ;',/,
     &'FOR STRAIGHT LINE OEPRECIATION AND A NEGATIVE INTEGER',/,
     &' FOR OOUBLE OECLINNG BALANCE WITH SWITCHING TO STRAIGHT',/,
.    &'LINE WHEN BENEFICIAL.  AN INTEGER IS REQUIREO.  EXAMPLE: -1')
      CALL FREAO(5, 'I V:', FLAG(1), 1)
C
C     INPUT THE REQUIREO RATES OF RETURN (PERCENT) WITH A NEGATIVE FLAG:
C
      WRITE(6,16)
16    FORMAT('ENTER  ROR1=REQUIREO RATES OF RETURN(PERCENT). ',/,
     &'UP TO 5 VALUES CAN BE ENTERED, IN ASSENOING OROER',/,
     &'. REAL VALUES ARE REQUIREO- I.E. VALUES MUST HAVE',/,
     &' A OECIMAL POINT.  THE LAST NUMBER MUST BE A NEGATIVE FLAG',/,
     &'FOR EXAMPLE:  10.0 12.0 15.0 -1.0')
      CALL FREAO(5, 'R V:', ROR1, 6)
C
C     OIVIOE THESE PERCENTAGES BY 100:
C
      OO 70 I = 1,6
      ROR(I) = ROR1(I) / 100.
70    CONTINUE
C
C     REAO IN THE POSSIBLE AVERAGE ORE GRAOES (MAX 6):
C
      WRITE(6,17)
17    FORMAT('ENTER AVEGRO=THE AVERAGE ORE GRAOE IN PERCENT U308',/,
     &' 6 VALUES MUST BE ENTEREO, WHICH MUST BE REAL, I.E. THEY MUST',/,
     &'CONTAIN A OECIMAL POINT.  AN EXAMPLE IS GIVEN BELOW:',/,
     &' 0.01 0.025 0.05 0.10 0.20 0.25')
      CALL FREAD(5, 'R V:', AVEGRO, 6)
C
C     REAO IN THE CORRESPONOING MILL RECOVERIES (MAX 6):
C
      WRITE(6,18)
18    FORMAT('ENTER MILREC=PERCENTAGE MILL RECOVERY',/,
     &' 6 VALUES ARE ENTEREO TO CORRESPONO TO ',/,
     &'THOSE VALUES ENTEREO IN AVEGRO',/,
     &'REAL VALUES ARE REQUIRED, AS SHOWN IN THE EXAMPLE BELOW:',/,
     &' 47.5 77.5 87.5 92.5 94.0 95.0')
      CALL FREAD(5, 'R V:', MILREC, 6)
C     INPUT THE PRICE ANO COST ESCALATION RATES, THEN DIVIOE BY 100:
C
      WRITE(6,19)
```

```
19    FORMAT('ENTER ESCALN=ESCALATION RATE IN PERCENT.',/,
     &' 4 VALUES ARE NEEDED, THE FIRST TWO ARE REVENUE ESCALATION',/,
     &'RATES IN PERIODS 1 AND 2, RESPECTIVELY.  THE THIRD AND FOURTH',/,
     &'VALUES ARE COST ESCALATION RATES IN PERIODS 1 AND 2 PERIODS',/,
     &'ALL VALUES ARE REAL.  AN EXAMPLE IS GIVEN BELOW:,',/,
     &' 10.0 7.0 6.0 5.0')
      CALL FREAD(5, 'R V:', ESCALN, 4)
C
      FLAG(5) = 0.
      DO 80 I = 1,4
      IF (ESCALN(I) .GT. 0.) FLAG(5) = 2.
      ESCALN(I) = 1. + ESCALN(I) / 100.
80    CONTINUE
C
C
C
C     LINE 100 IS RETURNED TO AUTOMATICALLY FROM LINE 700 IN ORDER
C  TO RERUN THE PROGRAM USING NEW INPUT DATA.
C
C
C
C
      WRITE(6,21)
21    FORMAT('ENTER FLAG(2), TPD, AND AVGRD RESPECTIVELY',/,
     &'   FLAG(2)=A NEGATIVE INTEGER FOR UNDERGROUND MINING AND ',/,
     &'           A POSITIVE INTEGER FOR OPEN PIT MINING',/,
     &'   TPD=MINE DAILY CAPACITY IN SHORT TONS PER CALANDER YEAR',/,
     &'   AVGRD=AVERAGE ORE GRADE IN PERCENT U308',/,
     &'FLAG(2) IS AN INTEGER WHILE TPD AND AVGRD ARE ',/,
     &'REAL NUMBERS.  AN EXAMPLE IS GIVEN BELOW:',/,
     &' 1 2000.0 0.05')
38    CALL FREAD(5, 'I V:', FLAG(2), 1, 'R:',TPD, 'R:', AVGRD)
      WRITE(6,987)
987   FORMAT('DRILLING INFORMATION IS REQUIRED.  IS YOUR',/,
     &'DRILLING COST DATA IN THE $ PER TON FORMAT',/,
     &'YES=1, NO=2')
      CALL FREAD(5, 'I:', IFOR)
      IF(IFOR.EQ.1) GO TO 383
      WRITE(6,986)
986   FORMAT('ENTER EXD, THE AVERAGE NUMBER OF FEET OF',/,
     &'EXPLORATION DRILLING DONE PER YEAR',/,
     &'NEXT ENTER THE AVERAGE NUMBER OF FEET OF DEVELOPMENT',/,
     &'DRILLING DONE PER YEAR- DEVD',/,
     &'LASTLY, ENTER CEXD, THE COST PER FOOT OF DRILLING AT',/,
     &'DATEC.  ACILIARY DRILLING COSTS ARE INCLUDED IN THIS',/,
     &'ESTIMATE.  FOR EXAMPLE:  20000. 20000. 32.')
      CALL FREAD(5, '3R:', EXD, DEVD, CEXD)
      WRITE(6,985)
```

```
985   FORMAT('THIS INFORMATION WILL BE USED TO CALCULATE THE',/,
     &'ANNUAL DRILLING COSTS AT THE GIVEN D/T RATIO OF',/,
     &'DRILLING WHICH WILL BE ADJUSTED FURTHER FDR DIFFERANT',/,
     &'D/T RATIOS FOR CALCULATING THE REQUIRED SELLING PRICE OF U')
C CALCULATE THE ANNUAL COST DF DRILLING FRDM DATEC TO DATEP
      DCOST=EXD * CEXD * (PARAM(4)/PARAM(3))
      GCDST=DEVD * CEXD * (PARAM(4)/PARAM(3))
383   WRITE(6,22)
22    FDRMAT('ENTER CDSTPT(10)=EXPLORATION, DEVELDPMENT ',/,
     &'AND OPERATING COSTS',/,
     &'SEE MANUAL FOR DETAILED EXPLANATION.  ALL VALUES ARE REAL.',/,
     &'AN EXAMPLE IS GIVEN BELDW:',/,
     &' 0.036 0.028 0.09 0.04 4.80 0.17 1.47 0.70 0.75 4.04')
      CALL FREAD(5, 'R V:', COSTPT, 10)
1010  CDNTINUE
C
C
C     SET ALL PRICE TO 0.
C
      DO 110 I = 1,5
      DO 110 J = 1,40
      PRICE(I,J) = 0.0
110   CONTINUE
C
C     ADJUST THE COST DATA UP FRDM THE DATE TAKEN (DATEC) TO THE NEW
C  DATE (DATEP) BY APPLYING THE PRICE INDICES.
C
C     APPLY THE MARSHALL & SWIFT MINING AND MILLING INDICES TO THE
C  CDSTS OF MINE PRIMARY DEVELOPMENT, PLANT & EQUIPMENT, AND MILL
C  CDNSTRUCTION:
C
      DO 120 I = 5,7
      COSTPT(I) = COSTPT(I) * PARAM(2) / PARAM(1)
120   CONTINUE
C
C     APPLY THE WHOLESALE PRICE INDICES FOR INDUSTRIAL CDMMDDITIES TD
C  THE REMAINING CDST DATA (COSTPT)
C
      DD 130 I = 1,4
      COSTPT(I) = COSTPT(I) * PARAM(4) / PARAM(3)
130   CDNTINUE
C
      DO 140 I = 8,10
      CDSTPT(I) = CDSTPT(I) * PARAM(4) / PARAM(3)
140   CONTINUE
C
C     MULTIPLY ALL CDST DATA BY TPD*365 TO GET THE EQUIVALENT
C  ANNUAL COST (ANCDST), WHERE TPD IS MINE DAILY CAPACITY AND 365
```

```
C  IS THE NUMBER OF DAYS IN A YEAR.
C
      DO 150 I = 1,10
      ANCOST(I) = COSTPT(I) * TPD * 365.
150   CONTINUE
C
      IF(IFOR.EQ.1) GO TO 153
      ANCOST(3) = DCOST
      ANCOST(4) = GCOST
C     MATCH THE APPROPRIATE MILL RECOVERY TO THE ORE GRADE (WHERE
C  'RECMIL' IS THE APPLICABLE MILL RECOVERY):
C
153   DO 160 I = 1,6
      IF (AVGRD .EQ. AVEGRD(I)) RECMIL = MILREC(I)
160   CONTINUE
C
      RECMIL = RECMIL / 100.
      AVGRD = AVGRD / 100.
C
C     SELECT EITHER THE SUBROUTINE APPROPRIATE FOR AN UNDERGROUND
C  OR OPEN PIT MINE:
C
180   IF (FLAG(2) .GT. 0.) GO TO 210
C
200   CALL UNDER (ANCOST, AVGRD, ESCALN, FLAG, MILDEP, PARAM,
     1RECMIL, ROR, TPD, DTREQ, MLIFE, NYEAR, PRICE,
     2TOTTON)
      GO TO 300
C
210   CALL OPEN (ANCOST, AVGRD, ESCALN, FLAG, MILDEP, PARAM,
     1RECMIL, ROR, TPD, DTREQ, MLIFE, NYEAR, PRICE,
     2TOTTON)
C
C     AFTER RETURN FROM THE SUBROUTINES WRITE THE RESULTS TO THE OUTPUT
C  FILE:
C
C
300   WRITE (7,305) FLAG(2), FLAG(3), FLAG(1)
305   FORMAT (I2,',',I2,',',I2)
C
      IF (FLAG(2) .GT. 0.0) GO TO 320
C
      WRITE (7,310) DATEP
310   FORMAT ('UNDERGROUND MINE AT  ',10A1)
      GO TO 340
C
320   WRITE (7,330) DATEP
330   FORMAT ('OPEN PIT MINE AT',6X,10A1)
```

```
C
340   WRITE (7,350) PARAM(2)
350   FDRMAT (2X,'MARSHALL & SWIFT INDEX (MINING & ',
     1'MILLING)',4X,F6.1)
C
      WRITE (7,360) PARAM(4)
360   FDRMAT (2X,'WHOLESALE PRICE INDEX (INDUSTRIALS)',
     110X,F6.1)
C
      WRITE (7,370) DATEC
370   FDRMAT (/,'INPUT COSTS VALID AT  ',10A1)
C
      WRITE (7,355) PARAM(1)
355   FORMAT (2X,'MARSHALL & SWIFT INDEX (MINING & ',
     1'MILLING)',4X,F6.1)
      WRITE (7,361) PARAM(3)
361   FORMAT (2X,'WHOLESALE PRICE INDEX (INDUSTRIALS)',
     110X,F6.1)
C
      WRITE (7,380) TDTTDN
380   FORMAT (/,'TOTAL ORE RESERVES (MILLION S.TON)',13X,F7.3)
C
      AVGRD  = AVGRD * 100.
      WRITE (7,390) AVGRD
390   FORMAT ('AVERAGE ORE GRADE (URANIUM DXIDE %)',14X,F5.3)
C
      WRITE (7,400) TPD
400   FORMAT ('DRE PRODUCTION RATE (TON PER CALENDAR DAY)',
     13X,F6.0)
C
      WRITE (7,405) MLIFE
405   FORMAT ('MINE DPERATING LIFE (YEARS)',21X,I2)
      LEADYR = NYEAR(6,1) - 1
      WRITE (7,407) LEADYR
407   FORMAT ('YEARS LEAD TIME, YEAR 0 TO START-UP',13X,I2)
C
      WRITE (7,410) NYEAR(10,1), NYEAR(10,2)
410   FDRMAT (/,'FIRST PERIOD PRICE AND CDST ESCALATION (YEARS)',2X,
     1I2,',',I3)
C
      DD 411 I = 1,4
411   ESC(I) = (ESCALN(I) - 1.) * 100.
C
      WRITE (7,412) (ESC(I), I = 1,4)
412   FDRMAT (2X,'PRICE ESCALATION RATES (PERCENT)',14X,F5.2,',',
     1F5.2,/,2X,'CDST  ESCALATION RATES (PERCENT)',14X,F5.2,',',F5.2)
C
      IF (FLAG(3) .GT. 0.0) GD TO 420
```

```
C
      WRITE (7,415)
415   FORMAT (/,'OTHER INCOME IS AVAILABLE FOR TAKING IMMEDIATE ',
     1'TAX BENEFITS',/,'AGAINST.')
C
      GO TO 435
C
420   WRITE (7,430)
430   FORMAT (/,'OTHER INCOME IS NOT AVAILABLE FOR TAKING IMMEDIATE ',
     1'TAX',/,'BENEFITS AGAINST.')
C
435   IF (FLAG(1) .LT. O.) GO TO 445
C
      WRITE (7,440)
440   FORMAT (/,'STRAIGHT LINE DEPRECIATION.')
C
      GO TO 450
C
445   WRITE (7,447)
447   FORMAT (/,'DOUBLE DECLINING BALANCE DEPRECIATION WITH ',
     1'SWITCHING TO',/,'STRAIGHT LINE.')
C
450   WRITE (7,455)
455   FORMAT (/,8X,'DEPTH TO',9X,'RATE OF RETURN',7X,
     1'REQUIRED PRICE',/,5X,'THICKNESS RATIO',7X,'(PERCENT)',
     212X,'$/LB U. OX.',/)
C
C     DO-LOOP TO PRINT OUT PRICE FIGURES:
C
460   DO 505 J = 1,40
      DO 500 I = 1,6
C
C     LEAVE DO-LOOP IF NEGATIVE DTREQ FLAG ENCOUNTERED. THEN
C  WRITE A NEGATIVE FLAG IN THE DTREQ COLUMN:
      IF (DTREQ(J) .LE. O.O) GO TO 510
C
C     LEAVE BLANK LINE AFTER EACH GROUP OF ROR'S BY USING THE
C  NEGATIVE ROR FLAG:
C
      IF (ROR1(I).GE. O.) GO TO 480
      WRITE (7,470)
470   FORMAT (' ')
C
      GO TO 505
C
480   WRITE (7,490) DTREQ(J), ROR1(I), PRICE(I,J)
490   FORMAT (9X,F5.1,15X,F5.2,15X,F6.2)
C
```

```
C     LEAVE THE DO-LOOP IF MAXIMUM PRICE LIMIT ENCOUNTERED
C  FOR ROR(1):
C
      IF (PRICE(1,J).GE.PARAM(11)) GO TO 530
C
500   CONTINUE
505   CONTINUE
C
C
510   WRITE (7,520)
520   FORMAT (10X,'-1.0')
C
      GO TO 700
C
530   WRITE (7,540)
540   FORMAT (52X,'-1.0')
C
C
C     READ IN FLAG TO DECIDE WHETHER TO RETURN FOR MORE FILES:
C
700   WRITE(6,23)
23    FORMAT('ENTER FLAG(4)=A POSITIVE INTEGER RERUNS PRICE2',/,
     &'WHILE A NEGATIVE INTEGER STOPS THE PROGRAM',/,
     &'FOR EXAMPLE  -1')
      CALL FREAD(5, 'I V:', FLAG(4), 1)
      IF (FLAG(4) .GT. 0.0)GO TO 38
C
      STOP
      END
C     SUBROUTINE OPEN
C     ---------------
C
C
C     THIS SUBROUTINE IS SIMILAR TO THE SUBROUTINE UNDER. IT:
C
C     (1) READS INTERACTIVELY DATA APPROPRIATE TO
C     UNDERGROUND MINING, INCLUDING:
C     (A) GENERAL DATA;
C     (B) TIME PERIODS FOR THE VARIOUS ACTIVITIES.
C
C     (2) ADJUSTS DRILLING AND EXPLORATION COSTS TO THE DEPTH TO
C     THICKNESS RATIO (D/T) OF THE OTHER INPUT DATA.
C
C     (3) ADJUSTS ALL COSTS TO AVERAGE ANNUAL COSTS FOR THEIR
C     RESPECTIVE TIME PERIODS.
C
C     (4) CALLS THE SUBROUTINE 'DEDUCT' AND IS RETURNED TO FROM IT.
C
```

```
C      (5) INITIATES THE DO-LOOP WHICH VARIES THE D/T AND ADJUSTS
C      THE COSTS TO THE RESPECTIVE D/T'S. YRCOST'S ARE AVERAGE ANNUAL
C      COSTS CORRECTED TO THEIR RESPECTIVE TIME PERIODS AND D/T.
C
C
       SUBROUTINE OPEN (ANCOST, AVGRD, ESCALN, FLAG, MILDEP, PARAM,
      1RECMIL, ROR, TPD, DTREQ, MLIFE, NYEAR,
      2PRICE, TOTTON)
C
C
       DIMENSION ANCOST(11), COSTES(35), DEPREC(35), DTREQ(40),
      1ESCALN(4), JYRPDV(2), KP(5), MPEQYR(5),
      2PARAM(12), PITDAT(11), PRICE(5,40),
      3REVESC(35), ROR(6), TAXCR(35), YRCOST(13), YRPDV(2)
       INTEGER FLAG(7)
       INTEGER IX(20)
       INTEGER NYEAR(10,2)
C
C      READ IN THE GENERAL DATA:
C
C
       WRITE(6,24)
24     FORMAT('ENTER OPEN PIT GENERAL PARAMETERS- ',/,
      &'FLAG(3) AND PITDAT(111)',/,
      &'   FLAG(3)=A NEGATIVE INTEGER WHEN EXTERNAL TAXABLE',/,
      &'       INCOME IS AVAILABLE; AND POSITIVE WHEN NONE',/,
      &'   PITDAT(11)=11 REAL VARIABLES FOR OPEN PIT OPERATIONS.',/,
      &'SEE MANUAL FOR FURTHER EXPLANATION.  EXAMPLE GIVEN BELOW:',/,
      &' -1 24. 100. 100. 100. 50. 50. 2. 6000. 3000. 269018. 576079.')
       CALL FREAD(5, 'I V:', FLAG(3), 1, 'R V:', PITDAT, 11)
C
C      READ IN THE REQUIRED DEPTH TO THICKNESS RATIOS:
C
       DO 43 I = 1,40
       DTREQ(I)=0.0
43     CONTINUE
       WRITE(6,25)
25     FORMAT('ENTER THE DEPTH/THICKNESS RATIOS FOR WHICH',/,
      &'THE PRICE IS REQUIRED.  8 REAL VARIABLES',/,
      &' DTREQ(1-8) ARE ENTERED ON THE FIRST LINE',/,
      &'THE VARIABLES ARE ORDERED FROM SMALLEST TO LARGEST',/,
      &'EXAMPLE: 10.0 15.0 20.0 25.0 30.0 35.0 40.0 45.0')
       CALL FREAD(5, 'R V:', DTREQ(1), 8)
       IF(DTREQ(8).LE.0.)GO TO 50
       WRITE(6,26)
26     FORMAT('ENTER DTREQ(9-16) AS SHOWN IN PREVIOUS EXAMPLE',/,
      &'IF FURTHER RATIOS ARE NOT REQUIRED ENTER 0.0')
       CALL FREAD(5, 'R V:', DTREQ(9), 8)
```

```
      IF(DTREQ(16).LE.O.)GO TO 50
      WRITE(6,27)
27    FORMAT('ENTER DTREQ(17-24) AS SHOWN IN PREVIOUS EXAMPLE',/,
     &'IF FURTHER RATIOS ARE NOT REQUIRED ENTER O.O')
      CALL FREAD(5, 'R V:', DTREQ(17), 8)
      IF(DTREQ(24).LE.O.)GO TO 50
      WRITE(6,28)
28    FORMAT('ENTER DTREQ(25-32) AS SHOWN IN PREVIOUS EXAMPLE',/,
     &'IF FURTHER RATIOS ARE NOT REQUIRED ENTER O.O')
      CALL FREAD(5, 'R V:', DTREQ(25), 8)
      IF(DTREQ(32).LE.O.)GO TO 50
      WRITE(6,29)
29    FORMAT('ENTER DTREQ(33-40) AS SHOWN IN PREVIOUS EXAMPLE',/,
     &'IF FURTHER RATIOS ARE NOT REQUIRED ENTER O.O')
      CALL FREAD(5, 'R V:', DTREQ(33), 8)
C
C
C     READ IN THE TIME PERIOD DATA:
C
C
50    WRITE(6,31)
31    FORMAT('ENTER NYEAR= EXPENDITURE YEARS INFORMATION.',/,
     &'ALL VALUES ENTERED ARE INTEGERS AND FOLLOW ',/,
     &'THE FOLLOWING SEQUENCE:',/,
     &'NYEAR(1,1) NYEAR(1,2) NYEAR(2,1) NYEAR(2,2),...,',/,
     &'NYEAR(10,1) NYEAR(10,2)',/,
     &' 1 2 1 2 1 3 2 4 6 7 8 17 5 5 18 18 11 20 7 7')
      CALL FREAD(5, 'I V:', IX, 20)
      K=O
      DO 888 L=1,10
      DO 888 M=1,2
      K=K+1
      NYEAR(L,M)=IX(K)
888   CONTINUE
C
      WRITE(6,32)
32    FORMAT('ENTER YRPDV=YEAR BEGINING AND YEAR ENDING OF MINE ',/,
     &'DEVELOPMENT.  2 REAL NUMBERS ARE ENTERED. SEE MANUAL.',/,
     &'EXAMPLE:  5.5 15.5')
      CALL FREAD(5, 'R V:', YRPDV, 2)
      WRITE(6,33)
33    FORMAT('ENTER MPEQYR=PURCHASE DATES OF PLANT AND EQUIPMENT',/,
     &'UP TO 5 VALUES CAN BE ENTERED, WITH THE LAST NUMBER ',/,
     &'BEING A NEGATIVE INTEGER WHICH ACTS AS A FLAG',/,
     &'NUMBERS ARE ENTERED FROM EARLIEST TO LATEST',/,
     &'FOR EXAMPLE:   6 10 13 -1')
      CALL FREAD(5, 'I V:', MPEQYR, 5)
C
```

```
C
C     COUNTER NYRS IS USED TO DETERMINE HOW MANY TIMES PLANT &
C  EQUIPMENT IS PURCHASED FOR THE MINE.
C
      DO 110 I = 1,5
      IF (MPEQYR(I).LE.0) GO TO 120
      NYRS = I
110   CONTINUE
C
C     DIVIDE PERCENTAGES AND CENTS BY 100:
C
120   DO 130 I = 2,6
      PITDAT(I) = PITDAT(I) / 100.
130   CONTINUE
C
C     ADJUST THE COSTS OF FIELD EXPENSE AND DRILLING TO THE DEPTH TO
C  THICKNESS RATIO OF THE OTHER DATA:
C
      KP(1) = 1
      KP(2) = 3
      KP(3) = 4
C
      DTADJ = 1. +((PITDAT(1) - PARAM(10)) / PARAM(10)) * PITDAT(2)
      DO 140 I = 1,3
      ANCOST(KP(I)) = DTADJ * ANCOST(KP(I))
140   CONTINUE
C
C     CALCULATE THE ANNUAL COST OF PREPARING THE ENVIRONMENTAL IMPACT
C  STATEMENT AND OBTAINING THE APPROPRIATE PERMITS:
C
      YRCOST(13) = PITDAT(10) * ALOG10(TPD) - PITDAT(11)
      YRCOST(13) = YRCOST(13) / (NYEAR(7,2) - NYEAR(7,1) + 1)
      WRITE(6,343)
343   FORMAT('ENTER NENV, THE NUMBER OF YEARS FROM 1977 TO',/,
     &'THE YEAR THE ENVIROMENTAL STUDY IS UNDERTAKEN')
      CALL FREAD(5, 'I:', NENV)
      DO 344 I=1,NENV
      YRCOST(13)=YRCOST(13)+(YRCOST(13)*0.10)
344   CONTINUE
C
C     DETERMINE THE MINE OPERATING LIFE:
C
      MLIFE = NYEAR(6,2) - NYEAR(6,1) + 1
C
C     ANNUAL BACKFILLING COST:
C
      ANCOST(11) = 9000000. * TPD * PITDAT(5) * PITDAT(6) / 500.
      ANCOST(11) = ANCOST(11) / (NYEAR(8,2) - NYEAR(8,1) + 1)
```

```
C
C     ANNUAL RECLAMATION COST:
C
      YRCOST(12) = 100. * TPD  * (PITDAT(8) + PITDAT(9) *
     1PITDAT(7)) / 500.
      YRCOST(12) = YRCOST(12) / (NYEAR(9,2) - NYEAR(9,1) + 1)
C
C  DETERMINE THE COST OF THE PLANT & EQUIPMENT PURCHASES IN
C  EACH OF THE PURCHASING YEARS:
C
      YRCOST(6) = ANCOST(6) * MLIFE / NYRS
C
C  CALCULATE THE FRACTIONS OF YEARS OF MINE PRIMARY
C  DEVELOPMENT AT BOTH THE BEGINNING AND END OF DEVELOPMENT:
C
      JYRPDV(1) = IFIX(YRPDV(1))
      JYRPDV(2) = IFIX(YRPDV(2))
      YFPDV = YRPDV(1) - JYRPDV(1)
      YLPDV = YRPDV(2) - JYRPDV(2)
      JYRPDV(1) = JYRPDV(1) + 1
      JYRPDV(2) = JYRPDV(2) + 1
C
C  TOTAL TONNAGE OF ORE IN MILLION SHORT TONS:
C
      TOTTON = TPD * 365. * MLIFE / 1000000.
C
C  ADJUST THE REMAINING ANCOST'S TO TRUE AVERAGE ANNUAL COSTS FOR
C  THEIR RESPECTIVE TIME PERIODS:
C
      DO 150 I = 1,4
      LL=(NYEAR(I,2) - NYEAR(I,1) + 1)
150   ANCOST(I) = ANCOST(I) * MLIFE /  LL
      YRCOST(2) = ANCOST(2)
      ANCOST(5) = ANCOST(5) * MLIFE / (YRPDV(2) - YRPDV(1))
      YRCOST(7) = ANCOST(7) * MLIFE / (NYEAR(5,2) - NYEAR(5,1) + 1)
      DO 160 I = 8,10
160   YRCOST(I) = ANCOST(I)
C
C  OPERATING COST = MINING + HAULAGE + MILLING:
C
      YRCOST(8) = YRCOST(8) + YRCOST(9) + YRCOST(10)
C
C
      CALL DEDUCT  (ANCOST, ESCALN, FLAG, MILDEP, MLIFE, MPEQYR,
     1NYEAR, NYRS, PARAM, YRCOST, YRPDV, COSTES, DEPREC,
     2LASTYR, REVESC, TAXCR)
C
C  THIS DO-LOOP IS THE MAIN ONE OF THE PROGRAM. IT PROVIDES
```

```
C  A NEW REQUIREO OEPTH TO THICKNESS RATIO (OT) ON EACH LOOP.
C
       NOT=O
200    OO 300 KOT = 1,40
C
       NOT = NDT + 1
C
C  THIS IF STATEMENT PROVIOES THE RETURN TO THE MAIN PROGRAM:
C
       IF (DTREQ(NOT).LE.O.O) GO TO 400
       DT = OTREQ(NDT)
C
C  AOJUST THE FOLLOWING COSTS TO THE NEW DEPTH TO THICKNESS RATIO:
C
C  (A) FIELO EXPENSE, EXPLORATION ORILLING, DEVELOPMENT DRILLING:
C
       OTAOJ = 1. + ((DT - PITOAT(1)) / PITOAT(1)) * PITOAT(2)
       DO 210 I = 1,3
       YRCOST(KP(I)) = DTAOJ * ANCOST(KP(I))
210    CONTINUE
C
C  (B) MINE PRIMARY DEVELOPMENT:
C
       YRCOST(5) = (1.+((DT-PITOAT(1))/PITOAT(1))*PITOAT(3))*ANCOST(5)
C
C  (C) BACKFILLING OF PIT:
C
       YRCOST(11)=(1.+((OT-PITDAT(1))/PITOAT(1))*PITDAT(4))*ANCOST(11)
C
C
       CALL INVEST (AVGRO, COSTES, OEPREC, OTREQ, FLAG, JYRPOV, KP,
      1LASTYR, MLIFE, MPEQYR, NDT, NYEAR, NYRS, PARAM, RECMIL,
      2REVESC, ROR, TAXCR, TPO, YFPOV, YLPOV, YRCOST, PRICE)
C
C  TEST TO DETERMINE IF THE PRICE FOR ROR(1) WAS EQUAL TO OR
C  GREATER THAN THE MAXIMUM ALLOWED:
C
       IF (PRICE(1,NOT) .GE. PARAM(11)) GO TO 400
C
300    CONTINUE
C
400    RETURN
C
       END
C  SUBROUTINE UNOER
C  ----------------
C
C
```

```
C   THIS SUBROUTINE IS SIMILAR TO THE SUBROUTINE OPEN.  IT:
C
C   (1) READS INTERACTIVELY DATA APPROPRIATE TO
C   UNDERGROUND MINING, INCLUDING:
C   (A) GENERAL DATA;
C   (B) TIME PERIODS FOR THE VARIOUS ACTIVITIES.
C
C   (2) ADJUSTS DRILLING AND EXPLORATION COSTS TO THE DEPTH TO
C   THICKNESS RATIO (D/T) OF THE OTHER INPUT DATA.
C
C   (3) ADJUSTS ALL COSTS TO AVERAGE ANNUAL COSTS FOR THEIR
C   RESPECTIVE TIME PERIODS.
C   (4) CALLS THE SUBROUTINE 'DEDUCT' AND IS RETURNED TO FROM IT.
C
C   (5) INITIATES THE DO-LOOP WHICH VARIES THE D/T AND ADJUSTS
C   THE COSTS TO THE RESPECTIVE D/T'S. YRCOST'S ARE AVERAGE ANNUAL
C   COSTS CORRECTED TO THEIR RESPECTIVE TIME PERIODS AND D/T.
C
C
       SUBROUTINE UNDER (ANCOST, AVGRD, ESCALN, FLAG, MILDEP, PARAM,
      1RECMIL, ROR, TPD, DTREQ, MLIFE, NYEAR,
      2PRICE, TOTTON)
C
C
       DIMENSION ANCOST(11), COSTES(35), DEPREC(35), DTREQ(40),
      1ESCALN(4), JYRPDV(2), KP(5), MPEQYR(5),
      2NYEAR(10,2), PARAM(12), PRICE(5,40), REVESC(35), ROR(6),
      3TAXCR(35), UGDATA(8), YRCOST(13), YRPDV(2)
       INTEGER FLAG(7)
       INTEGER IY(14)
       INTEGER IZ(2)
C
C   READ IN THE GENERAL DATA:
C
C
       WRITE(6,34)
34     FORMAT('ENTER UNDERGROUND GENERAL PARAMETERS- ',/,
      &'FLAG(3) AND UGDATA(8)',/,
      &'   FLAG(3)=A NEGATIVE INTEGER WHEN EXTERNAL TAXABLE',/,
      &'       INCOME IS AVAILABLE; AND POSITIVE WHEN NONE',/,
      &'   UGDATA(8)=8 REAL VARIABLES FOR UNDERGROUND OPERATIONS.',/,
      &'SEE MANUAL FOR FURTHER EXPLANATION.  EXAMPLE GIVEN BELOW:',/,
      &' -1 76.0 100.0 100.0 12.0 5.0 179345.0 384053.0 25.0')
       CALL FREAD(5, 'I V:', FLAG(3), 1, 'R V:', UGDATA, 8)
C
C   READ IN THE REQUIRED DEPTH TO THICKNESS RATIOS:
C
       WRITE(6,25)
```

```
25    FDRMAT('ENTER THE DEPTH/THICKNESS RATIDS FOR WHICH',/,
     &'THE PRICE IS REQUIRED.  DRDER FRDM SMALLEST TO LARGEST',/,
     &' 8 REAL VARIABLES DTREQ(1-8) ARE INPUTTED DN THE FIRST LINE',/,
     &'EXAMPLE: 10.0 15.0 20.0 25.0 30.0 35.0 40.0 45.0')
      DO 43 I=1,40
      DTREQ(I)=0.0
43    CDNTINUE
      CALL FREAD(5, 'R V:', DTREQ(1), 8)
      IF(DTREQ(8).LE.O.)GO TD 50
      WRITE(6,26)
26    FORMAT('ENTER DTREQ(9-16) IF REQUIRED, AS SHDWN IN PREVIDUS ',/,
     &'EXAMPLE IF FURTHER RATIDS ARE NDT REQUIRED ENTER 0.0')
      CALL FREAD(5, 'R V:', DTREQ(9), 8)
      IF(DTREQ(16).LE.O.)GO TD 50
      WRITE(6,27)
27    FDRMAT('ENTER DTREQ(17-24) IF REQUIRED, AS SHDWN IN PREVIDUS ',/,
     &'EXAMPLE IF FURTHER RATIOS ARE NDT REQUIRED ENTER 0.0')
      CALL FREAD(5, 'R V:', DTREQ(17), 8)
      IF(DTREQ(24).LE.O.)GD TD 50
      WRITE(6,28)
28    FORMAT('ENTER DTREQ(25-32) IF REQUIRED, AS SHDWN IN PREVIOUS ',/,
     &' EXAMPLE IF FURTHER RATIDS ARE NDT REQUIRED ENTER 0.0')
      CALL FREAD(5, 'R V:', DTREQ(25), 8)
      IF(DTREQ(32).LE.O.)GD TD 50
      WRITE(6,29)
29    FDRMAT('ENTER DTREQ(33-40) IF REQUIRED, AS SHOWN IN PREVIDUS ',/,
     &'EXAMPLE IF FURTHER RATIDS ARE NDT REQUIRED ENTER 0.0')
      CALL FREAD(5, 'R V:', DTREQ(33), 8)
C
C
C  READ IN THE TIME PERIDD DATA:
C
C
50    WRITE(6,31)
31    FORMAT('ENTER NYEAR=EXPENDITURE YEARS INFDRMATIDN. ',/,
     &'ALL VALUES ARE INTEGERS AND ENTERED IN THE FDLLDWING SEQUENCE',/,
     &'NYEAR(1,1) NYEAR(1,2) NYEAR(2,1) NYEAR(2,2),....,',/,
     &'NYEAR(7,1) NYEAR(7,2).',/,
     &'FDR EXAMPLE: 1 2 1 2 1 3 2 4 6 7 8 17 5 5 ')
      CALL FREAD(5, 'I V:', IY, 14)
      K=0
      DD 777 J=1,7
      DD 777 I=1,2
      K=K+1
      NYEAR(J,I)=IY(K)
777   CONTINUE
      WRITE(6,35)
35    FDRMAT('ENTER NYEAR(10,1) AND NYEAR(10,2). THESE ARE THE ENDS',/,
```

```
      &'OF THE FIRST PERIODS OF PRICE AND COST ESCALATION.  IF NO',/,
      &'ESCALATION IS APPLIED ENTER 0 FOR EACH VALUE.  ',/,
      &'FOR EXAMPLE:  7 7')
       CALL FREAD(5, 'I V:', IZ, 2)
       K=0
       DO 666 I=1,2
       K=K+1
       NYEAR(10,I)=IZ(K)
666    CONTINUE
C
C  SET VALUE FOR NYEARS (I = 8,9) TO 0:
C
       DO 70 I = 8,9
       DO 70 J = 1,2
       NYEAR(I,J) = 0
70     CONTINUE
C
       WRITE(6,32)
32     FORMAT('ENTER YRPDV=YEAR BEGINING AND YEAR ENDING OF MINE ',/,
      &'DEVELOPMENT.  2 REAL NUMBERS ARE ENTERED. SEE MANUAL.',/,
      &'EXAMPLE:  5.5 15.5')
       CALL FREAD(5, 'R V:', YRPDV, 2)
C
       WRITE(6,33)
33     FORMAT('ENTER MPEQYR=MINE PLANT AND EQUIPMENT PURCHASE DATES',/,
      &'UP TO 5 NUMBERS ARE ENTERED , THE LAST BEING A NEGATIVE FLAG',/,
      &'NUMBERS ARE ENTERED FROM EARLIEST TO LATEST.  ',/,
      &'FOR EXAMPLE:  6,10,13,-1')
       CALL FREAD(5, 'I V:', MPEQYR, 5)
C
C
C  COUNTER 'NYRS' IS USED TO DETERMINE HOW MANY TIMES PLANT &
C  EQUIPMENT IS PURCHASED FOR THE MINE.
C
       DO 110 I = 1,5
       IF (MPEQYR(I).LE.0) GO TO 120
       NYRS = I
110    CONTINUE
C
C  DIVIDE PERCENTAGES BY 100:
C
120    DO 130 I = 2,5
       UGDATA(I) = UGDATA(I) / 100.
130    CONTINUE
       UGDATA(8) = UGDATA(8) / 100.
C
C  ADJUST COSTS OF FIELD EXPENSE AND DRILLING TO THE DEPTH
C  TO THICKNESS RATIO OF THE OTHER DATA:
```

```
C
      KP(1) = 1
      KP(2) = 3
      KP(3) = 4
C
      DTADJ = 1. + ((UGDATA(1) - PARAM(10)) / PARAM(10)) * UGDATA(2)
C
      DO 140 I = 1,3
      ANCOST(KP(I)) = DTADJ * ANCOST(KP(I))
140   CONTINUE
C
C  INCREASE MINING COST BY 'MESA' PROPORTION (I.E. THE AMOUNT
C  OF ADDITIONAL COST BROUGHT ON BY THE RECENT STRICT SUPERVISION
C  BY THE 'MINE ENVIRONMENTAL SAFETY ADMINISTRATION').
C
      ANCOST(8) = ANCOST(8) * (1. + UGDATA(5))
C
C  CALCULATE THE ANNUAL COST OF PREPARING THE ENVIRONMENTAL IMPACT
C  STATEMENT AND OBTAINING THE APPROPRIATE PERMITS:
C
      YRCOST(13) = UGDATA(6) * ALOG10(TPD) - UGDATA(7)
      YRCOST(13) = YRCOST(13) / (NYEAR(7,2) - NYEAR(7,1) + 1)
      WRITE(6,343)
343   FORMAT('ENTER NENV, THE NUMBER OF YEARS FROM 1977 TO',/,
     &'THE YEAR THE ENVIROMENTAL STUDY IS UNDERTAKEN')
      CALL FREAD(5, 'I:', NENV)
      DO 344 I=1,NENV
      YRCOST(13)=YRCOST(13)+(YRCOST(13)*0.10)
344   CONTINUE
C
C  DETERMINE MINE OPERATING LIFE:
C
      MLIFE = NYEAR(6,2) - NYEAR(6,1) + 1
C  DETERMINE THE COST OF THE PLANT & EQUIPMENT PURCHASES IN
C  EACH OF THE PURCHASING YEARS:
C
      YRCOST(6) = ANCOST(6) * MLIFE / NYRS
C
C  CALCULATION OF FRACTIONS OF YEARS OF MINE PRIMARY DEVELOPMENT
C  AT BOTH THE START AND END OF PRIMARY DEVELOPMENT:
C
      JYRPDV(1) = IFIX(YRPDV(1))
      JYRPDV(2) = IFIX(YRPDV(2))
      YFPDV = YRPDV(1) - JYRPDV(1)
      YLPDV = YRPDV(2) - JYRPDV(2)
      JYRPDV(1) = JYRPDV(1) + 1
      JYRPDV(2) = JYRPDV(2) + 1
C
```

```
C   THIS IF STATEMENT STOPS ANY OETAILEO CALCULATION IN THE SUBROUTINE
C   INCOME CAUSEO BY ROUNO OFF ERROR WHEN THE MINE PRIMARY OEVELOPMENT
C   STOPS AT MINE START-UP.
C
        IF (YLPOV.LE.0.001) YLPOV = 0.0
C
C   TOTAL TONNAGE (MILLION S.TONS) OF ORE IN THE GROUNO, INCLUOING
C   PILLAR REQUIREMENTS:
C
        TOTTON = TPD * 365. * MLIFE / ((1. - UGOATA(8)) * 1000000.)
C
C   AOJUST THE REMAINING ANCOST'S TO TRUE AVERAGE ANNUAL COSTS FOR
C   THEIR RESPECTIVE TIME PERIODS:
C
        DO 150 I = 1,4
150     ANCOST(I) = ANCOST(I) * MLIFE / (NYEAR(I,2) - NYEAR(I,1) + 1)
        YRCOST(2) = ANCOST(2)
        ANCOST(5) = ANCOST(5) * MLIFE / (YRPOV(2) - YRPDV(1))
        YRCOST(7) = ANCOST(7) * MLIFE / (NYEAR(5,2) - NYEAR(5,1) + 1)
        YRCOST(9) = ANCOST(9)
        YRCOST(10) = ANCOST(10)
        YRCOST(11) = 0.
        YRCOST(12) = 0.
C
C
        CALL DEOUCT (ANCOST, ESCALN, FLAG, MILOEP, MLIFE, MPEQYR,
       1NYEAR, NYRS, PARAM, YRCOST, YRPDV, COSTES, OEPREC,
       2LASTYR, REVESC, TAXCR)
C
C   THIS OO-LOOP IS THE MAIN ONE OF THE PROGRAM. IT PROVIDES
C   A NEW REQUIREO DEPTH TO THICKNESS RATIO (OT) ON EACH LOOP.
C
        NOT=0
200     DO 300 KOT = 1,40
C
        NDT = NOT + 1
C   THIS STATEMENT PROVIOES THE RETURN TO THE MAIN PROGRAM:
C
        IF (DTREQ(NOT).LE.0.0) GO TO 400
C
        OT= OTREQ(NOT)
C
C   AOJUST THE FOLLOWING COSTS TO THE NEW DEPTH TO THICKNESS RATIO:
C
C   (A) FIELD EXPENSE, EXPLORATION ORILLING, OEVELOPMENT DRILLING:
C
        OTAOJ = 1. + ((DT - UGDATA(1)) / UGDATA(1)) * UGOATA(2)
        OO 210 I = 1,3
```

```
          YRCOST(KP(I)) = DTADJ * ANCOST(KP(I))
  210     CONTINUE
C
C   (B) MINE PRIMARY DEVELOPMENT:
C
          YRCOST(5) = (1.+((DT-UGDATA(1))/UGDATA(1))*UGDATA(3))*ANCOST(5)
C
C   (C) MINING:
C
          YRCOST(8) = (1.+((DT-UGDATA(1))/UGDATA(1))*UGDATA(4))*ANCOST(8)
C
C   OPERATING COST = MINING + HAULAGE + MILLING:
C
          YRCOST(8) = YRCOST(8) + YRCOST(9) + YRCOST(10)
C
C
          CALL INVEST (AVGRD, COSTES, DEPREC, DTREQ, FLAG, JYRPDV, KP,
         1LASTYR, MLIFE, MPEQYR, NDT, NYEAR, NYRS, PARAM, RECMIL,
         2REVESC, ROR, TAXCR, TPD, YFPDV, YLPDV, YRCOST, PRICE)
C
C   ON RETURN FROM THE SUBROUTINES TEST TO DETERMINE IF THE PRICE
C   FOR THE ROR(1) WAS EQUAL TO OR GREATER THAN THE MAXIMUM ALLOWED:
C
          IF (PRICE(1,NDT) .GE. PARAM(11)) GO TO 400
C
  300     CONTINUE
C
  400     RETURN
C
          END
C   SUBROUTINE DEDUCT
C   -----------------
C
C
C   THIS SUBROUTINE CALCULATES FOR EACH YEAR:
C
C   (1) THE ESCALATION FACTORS APPLICABLE TO BOTH REVENUES
C   (REVESC) AND COSTS (COSTES);
C
C   (2) THE INVESTMENT TAX CREDIT (TAXCR);
C
C   (3) THE DEPRECIATION (DEPREC).
C
C
          SUBROUTINE DEDUCT (ANCOST, ESCALN, FLAG, MILDEP, MLIFE, MPEQYR,
         1NYEAR, NYRS, PARAM, YRCOST, YRPDV, COSTES, DEPREC,
         2LASTYR, REVESC, TAXCR)
C
```

```
      DIMENSION ANCOST(11), COSTES(35), DEPREC(35), DPCMIL(35),
     1DPCPEQ(35), ESCALN(4), LIFE(4), MPEQYR(5),
     2NYEAR(10,2), PARAM(12), REVESC(35), TAXCR(35),
     3YRCOST(13), YRPDV(2)
      INTEGER FLAG(7)
C
C  A POSITIVE FLAG(6) INDICATES THAT KLEMENIC'S EXPLORATION TIME
C  PERIODS ARE BEING USED BEGINNING AT THE END OF HIS YEAR 3. IT
C  THEN CAUSES THE PAST 3 YEAR'S COSTS TO BE TAKEN INTO ACCOUNT IN
C  TAXATION DEDUCTION CALCULATIONS IN THE SUBROUTINE 'INVEST'.
C
      FLAG(6) = 0.
      IF ((NYEAR(1,2) - NYEAR(1,1) .EQ. 1) .AND. (NYEAR(2,2) -
     1NYEAR(2,1) .EQ. 1) .AND. (NYEAR(3,2) - NYEAR(3,1) .EQ. 2))
     2FLAG(6) = 2.
C
C  FIND THE PROJECT FINAL YEAR (LASTYR):
C
      LASTYR = NYEAR(9,2)
      IF (NYEAR(6,2) .GT. NYEAR(9,2)) LASTYR = NYEAR(6,2)
C
C  SET COSTES AND REVESC TO 1.0 AND DEPREC AND TAXCR TO 0.:
C
      DO 10 I = 1,35
      COSTES(I) = 1.
      DEPREC(I) = 0.
      REVESC(I) = 1.
      TAXCR(I) = 0.
 10   CONTINUE
C
C  IF THE INDICATORS ARE APPROPRIATE, GO TO THE SIMPLIFIED STRAIGHT
C  LINE METHOD FOR THE UNDERGROUND CASE. A POSITIVE FLAG(7) INDICATES
C  THAT THE SIMPLIFIED METHOD IS BEING USED.
C
      FLAG(7) = 0.
      Y = FLOAT(NYEAR(6,1))
      IF ((FLAG(2) .LT. 0.) .AND. (FLAG(1) .GT. 0.) .AND. (FLAG(5)
     1.EQ. 0.) .AND. (PARAM(8) .EQ. 0.) .AND. (YRPDV(2) - 0.001
     2.LT. Y - 1.)) GO TO 600
C
C  DECIDE IF ESCALATION IS REQUIRED AT ALL:
C
      IF (FLAG(5) .EQ. 0.) GO TO 100
C
C  DECIDE IF PRICE ESCALATION IS REQUIRED:
C
      IF (NYEAR(10,1) .EQ. 0) GO TO 50
C
```

```
C   CALCULATE THE PRICE ESCALATIDN FACTORS FOR THE FIRST AND THEN
C   THE SECOND PERIOD:
C
       N = NYEAR(10,1)
       IF (N .GT. LASTYR) N = LASTYR
       REVESC(1) = ESCALN(1)
       DO 20 I = 2,N
20     REVESC(I) = REVESC(I-1) * ESCALN(1)
C
       IF (N .EQ. LASTYR) GD TO 50
       N = N + 1
       DO 30 I = N, LASTYR
30     REVESC(I) = REVESC(I-1) * ESCALN(2)
C
C   DECIDE IF COST ESCALATION IS REQUIRED:
C
50     IF (NYEAR(10,2) .EQ. 0) GO TO 100
C
C   CALCULATE THE COST ESCALATION FACTORS FOR THE FIRST AND THEN
C   THE SECOND PERIOD:
C
       N = NYEAR(10,2)
       IF (N .GT. LASTYR) N = LASTYR
       COSTES(1) = ESCALN(3)
       DO 60 I = 2,N
60     COSTES(I) = COSTES(I-1) * ESCALN(3)
C
       IF (N .EQ. LASTYR) GO TO 100
       N = N + 1
       DO 70 I = N, LASTYR
70     CDSTES(I) = COSTES(I-1) * ESCALN(4)
C
C   DETERMINE THE USEFUL LIFE OF THE PLANT & EQUIPMENT (P&E) FOR EACH
C   PURCHASE. THIS IS USED FOR DETERMINING THE INVESTMENT TAX CREDIT
C   AND DEPRECIATION. IT IS ASSUMED THAT NO P&E IS PURCHASED BEFORE THE
C   FIRST DEVELOPMENT YEAR.
C
100    DO 110 I = 1,4
110    LIFE(I) = 0
C
       IF (NYRS .EQ. 1) GO TO 125
C
       DO 120 I = 2, NYRS
120    LIFE(I-1) = MPEQYR(I) - MPEQYR(I-1)
125    LIFE(NYRS) = NYEAR(6,2) - MPEQYR(NYRS) + 1
C
C   CALCULATE THE INVESTMENT TAX CREDIT FOR BDTH THE MILL AND P&E:
C
```

```
      IF (PARAM(8) .EQ. O.) GO TO 200
C
C  (A) MILL (CRMILL = MILL TAX CREDIT):
C
      CRMILL = O.
      IQT=NYEAR(5,2)
      IQR=NYEAR(5,1)
      DO 130 I = IQR, IQT
130   CRMILL = YRCOST(7) * COSTES(I) * PARAM(8) * PARAM(9)
     1+ CRMILL
C
C  (B) MINE P&E:
C
C  BASED ON THE USEFUL LIFE THE FRACTION OBTAINED OF THE MAXIMUM
C  PERCENTAGE ALLOWABLE IS CALCULATED.
C
      DO 140 I = 1,NYRS
      FRAC = 1.
      IF (LIFE(I).LE.6) FRAC = 2. / 3.
      IF (LIFE(I).LE.4) FRAC = 1. / 3.
      TAXCR(MPEQYR(I)) = YRCOST(6) * COSTES(MPEQYR(I)) * FRAC
     1* PARAM(8)
140   CONTINUE
C
C  ALLOCATE THE INVESTMENT TAX CREDIT DUE TO THE MILL TO THE FIRST
C  OPERATING YEAR:
C
      TAXCR(NYEAR(6,1)) = TAXCR(NYEAR(6,1)) + CRMILL
C
C
C  STRAIGHT LINE DEPRECIATION:
C
200   IF (FLAG(1) .LT. O.) GO TO 300
C
C  (A) MILL (DEPMIL = ANNUAL MILL DEPRECIATION):
C
C  THE DEPRECIATION IS ACCUMULATED FOR DEDUCTION WHEN MINING BEGINS.
C
      DEPMIL = O.
      DO 210 I = IQR, IQT
210   DEPMIL = YRCOST(7) * COSTES(I) + DEPMIL
C
      DEPMIL = DEPMIL / MILDEP
C
C  (B) MINE P&E (DEPPEQ = ANNUAL P&E DEPRECIATION):
C
      DO 220 I = 1, NYRS
      DEPPEQ = YRCOST(6) * COSTES(MPEQYR(I)) / LIFE(I)
```

```
      N = MPEQYR(I) - 1
      IQQ=LIFE(I)
      DO 220 J = 1, IQQ
      DEPREC(N+J) = DEPPEQ
220   CONTINUE
C
C  SUM THE MILL AND MINE P&E DEPRECIATION OVER THE OPERATING YEARS:
C
      N = NYEAR(6,2) - NYEAR(6,1) + 1
      K = NYEAR(6,2)
      IF (MILDEP .LT. N) K = NYEAR(6,1) + MILDEP - 1
C
      IQY=NYEAR(6,1)
      DO 230 I = IQY, K
230   DEPREC(I) = DEPREC(I) + DEPMIL
C
      GO TO 500
C
C  DOUBLE DECLINING BALANCE DEPRECIATION WITH SWITCHING TO STRAIGHT
C  LINE WHEN BENEFICIAL:
C
C  (A) MILL (DPCMIL = ANNUAL MILL DEPRECIATION):
C
300   DO 310 I = 1,35
310   DPCMIL(I) = 0.
C
C  CALCULATE THE TOTAL MILL COST (COSTML):
C
      COSTML = 0.
      DO 320 I = IQR, IQT
320   COSTML = YRCOST(7) * COSTES(I) + COSTML
C
C  DETERMINE THE ANNUAL DEPRECIATION OVER THE OPERATING YEARS:
C
      N = NYEAR(6,2) - NYEAR(6,1) + 1
      K = NYEAR(6,2)
      IF (MILDEP .LT. N) K = NYEAR(6,1) + MILDEP - 1
C
      IQY=NYEAR(6,1)
      DO 330 I = IQY, K
      J = I
      M = MILDEP - (I - NYEAR(6,1))
      COSTML = COSTML - DPCMIL(I-1)
C
C  CHECK IF STRAIGHT LINE DEPRECIATION IS THE GREATEST AND ACCEPT IT
C  IF IT IS:
C
      DEPMIL = COSTML / M
```

```
      DPCMIL(I) = COSTML * 2. / MILDEP
      IF (DEPMIL .GE. DPCMIL(I)) GO TO 340
330   CONTINUE
C
      GO TO 400
C
340   DO 350 I = J, K
350   DPCMIL(I) = DEPMIL
C
C  (B) P&E (DPCPEQ = P&E ANNUAL DEPRECIATION):
C
400   DO 410 I = 1,35
410   DPCPEQ(I) = O.
C
      DO 440 I = 1,NYRS
      COSTPE = YRCOST(6) * COSTES(MPEQYR(I))
      IQW=LIFE(I)
      DO 420 K = 1, IQW
      J = K
      M = IQW - K + 1
      N = MPEQYR(I) + K - 1
      IF (K .EQ. 1) GO TO 415
      COSTPE = COSTPE - DPCPEQ(N-1)
C
C  ACCEPT STRAIGHT LINE DEPRECIATION IF IT IS GREATEST:
C
415   DEPPEQ = COSTPE / M
      DPCPEQ(N) = COSTPE * 2. / LIFE(I)
      IF (OEPPEQ .GE. OPCPEQ(N)) GO TO 430
420   CONTINUE
C
      GO TO 440
C
430   DO 439 L = J, IQW
      N = MPEQYR(I) + L - 1
      DPCPEQ(N) = DEPPEQ
439   CONTINUE
440   CONTINUE
C
C  SUM THE MILL AND P&E DEPRECIATION:
C
      DO 450 I = 1, LASTYR
450   DEPREC(I) = DPCMIL(I) + DPCPEQ(I)
C
C  IF THERE IS NO INCOME AVAILABLE AGAINST WHICH TO EXPENSE THE
C  DEVELOPMENT YEAR EXPENDITURES, THEN SUM THE INVESTMENT TAX CREDIT
C  FOR THESE YEARS INTO THE FIRST OPERATING YEAR:
C
```

```
500   IF ((FLAG(3) .LT. O.) .OR. (PARAM(8) .EQ. O.)) GO TO 520
      N = NYEAR(6,1) - 7
      IF (N.LE.O) N = 1
      J = NYEAR(6,1) - 1
      OO 510 I = N, J
      TAXCR(NYEAR(6,1)) = TAXCR(NYEAR(6,1)) + TAXCR(I)
510   CONTINUE
C
520   RETURN
C
C
C  SIMPLIFIEO STRAIGHT LINE DEPRECIATION:
600   FLAG(7) = 2.
C
C  PLANT & EQUIPMENT OEPRECIATION:
C
      DEPPEQ = ANCOST(6) * MLIFE / (NYEAR(6,2) - MPEQYR(1) + 1)
C
C  MILL OEPRECIATION:
C
      M = MILDEP
      IF (MILOEP .LT. MLIFE) M = MLIFE
      OEPMIL = ANCOST(7) * MLIFE / M
C
C  ALLOCATE TO YEARS:
C
      IQI=NYEAR(6,2)
      IQU=MPEQYR(1)
      DO 610 I = IQU, IQI
610   DEPREC(I) = OEPPEQ
C
      DO 620 I = IQY, IQI
620   DEPREC(I) = DEPREC(I) + OEPMIL
C
      RETURN
C
      ENO
C  SUBROUTINE INVEST
C  -----------------
C
C
C  THIS SUBROUTINE ALLOCATES ALL COSTS TO THEIR APPLICABLE YEARS,
C  ESCALATES THEM IF NECESSARY ANO CALCULATES THE ANNUAL INVESTMENT
C  CASH FLOWS. IT ALSO OETERMINES THE APPLICABLE COST DEPLETION
C  ALLOWANCE ANO THE TOTAL EXPLORATION COST WRITE-OFF TO BE OEOUCTEO
C  FROM THE DEPLETION ALLOWANCE. IT THEN CALLS THE SUBROUTINE 'NETPV'.
C
C
```

```
      SUBROUTINE INVEST (AVGRD, COSTES, DEPREC, DTREQ, FLAG, JYRPDV,
     1KP, LASTYR, MLIFE, MPEQYR, NDT, NYEAR, NYRS, PARAM,
     2RECMIL, REVESC, ROR, TAXCR, TPD, YFPDV, YLPDV, YRCOST,
     3PRICE)
C
      DIMENSION COST(13,35), CFINV(35), COSTES(35), DEPREC(35),
     1DTREQ(40), JYRPDV(2), KP(5), MPEQYR(5),
     2NYEAR(10,2), PARAM(12), PRICE(5,40), REVESC(35), ROR(6),
     3TAXCR(35), YRCOST(13)
      INTEGER FLAG(7)
C
C  RESET THE ANNUAL COST MATRIX COST(13,35) AND THE ANNUAL
C  INVESTMENT CASH FLOWS CFINV(35) TO 0.
C
      DO 10 J = 1,35
      CFINV(J) = 0.0
      DO 10 I = 1,13
      COST(I,J) = 0.0
10    CONTINUE
C
C  FOR A NEGATIVE FLAG(3), TAKE THE TAX BENEFIT OF EXPENSING
C  INTANGIBLE FIELD EXPENSE, DRILLING AND BACKFILLING
C  EXPENDITURES AGAINST OTHER COMPANY INCOME IN THE NON-OPERATING
C  YEARS.
C
      IF (FLAG(3) .GT. 0.0) GO TO 30
C
      KP(4) = 11
      KP(5) = 13
C
      EXTAX = 1. - PARAM(7)
C
      DO 20 I = 1,5
      YRCOST(KP(I)) = YRCOST(KP(I)) * EXTAX
20    CONTINUE
C
C  ALLOCATE COSTS TO THEIR APPLICABLE YEARS:
C
C  (A) FIELD EXPENSE, PROPERTY ACQUISITION, EXPLORATION AND
C  DEVELOPMENT DRILLING:
C
30    DO 40 I = 1,4
      IQJ=NYEAR(I,1)
      IQK=NYEAR(I,2)
      DO 40 J = IQJ, IQK
      COST(I,J) = YRCOST(I)
40    CONTINUE
C
```

```
C  (B) MILL CONSTRUCTION:
C
      IQT=NYEAR(5,2)
      IQR=NYEAR(5,1)
      DO 50 J = IQR, IQT
      COST(7,J) = YRCOST(7)
50    CONTINUE
C
C  (C) RECLAMATION IF OPEN PIT:
C
      IF (FLAG(2) .LT. 0.0) GO TO 60
      IQO=NYEAR(9,1)
      IQE=NYEAR(9,2)
      DO 59 J = IQO, IQE
      COST(12,J) = YRCOST(12)
59    CONTINUE
60    CONTINUE
C
C  (D) MINE PRIMARY DEVELOPMENT:
C
C  1) FIRST YEAR COST:
C
      COST(5,JYRPDV(1)) = YRCOST(5) * (1. - YFPDV)
C
C  2) FINAL YEAR COST:
C
      COST(5,JYRPDV(2)) = YRCOST(5) * YLPDV
C  3) INTERMEDIATE YEARS:
C
      M = JYRPDV(2) - 1
      N = JYRPDV(1) + 1
      IF (N .GT. M) GO TO 71
      DO 70 J = N, M
      COST(5,J) = YRCOST(5)
70    CONTINUE
C
C  IF OTHER INCOME IS AVAILABLE EXPENSE PRIMARY DEVELOPMENT COSTS
C  BEFORE START-UP:
C
71    IF (FLAG(3) .GT. 0.0) GO TO 91
C
      N = NYEAR(6,1) - 1
      IQP=JYRPDV(1)
      DO 80 J = IQP, N
      COST(5,J) = COST(5,J) * EXTAX
80    CONTINUE
C
C  SIMILARLY, EXPENSE THE RECLAMATION COSTS FROM AFTER MINING
```

```
C  CEASES:
C
      IF (FLAG(2) .LT. 0.0) GO TO 91
C
      N = NYEAR(6,2) + 1
      IQE=NYEAR(9,2)
      DO 90 J = N, IQE
      COST(12,J) = COST(12,J) * EXTAX
90    CONTINUE
C
C  (E) PLANT & EQUIPMENT:
C
91    DO 100 N = 1,NYRS
      J = MPEQYR(N)
      COST(6,J) = YRCOST(6)
100   CONTINUE
C
C  (F) OPERATING COST:
C
      IQY=NYEAR(6,1)
      IQI=NYEAR(6,2)
      DO 120 I = 8,10
      DO 120 J = IQY, IQI
      COST(I,J) = YRCOST(I)
120   CONTINUE
C
C  (G) ENVIRONMENTAL IMPACT STATEMENT:
C
      IQA=NYEAR(7,1)
      IQS=NYEAR(7,2)
      DO 130 J = IQA, IQS
130   COST(13,J) = YRCOST(13)
C
C  (H) BACKFILLING:
C
      IF (FLAG(2) .LT. 0.) GO TO 150
      IQD=NYEAR(8,1)
      IQF=NYEAR(8,2)
      DO 140 J = IQD, IQF
140   COST(11,J) = YRCOST(11)
C
C
C  ESCALATE ALL COSTS:
C
150   IF (FLAG(5) .EQ. 0.) GO TO 161
      DO 160 J = 1,LASTYR
      DO 160 I = 1,13
      COST(I,J) = COST(I,J) * COSTES(J)
```

```
160   CONTINUE
C
C
C  SUM THE INVESTMENTS TO GET THE ANNUAL INVESTMENT CASH
C  FLOWS (CFINV(J)):
C
C  (A) FOR YEAR 1 TO THE YEAR BEFORE START-UP:
C
      N = NYEAR(6,1) - 1
161    DO 210 J = 1,N
      DO 210 I = 1,7
      CFINV(J) = CFINV(J) + COST(I,J)
210   CONTINUE
C
C  ADDITION OF WORKING CAPITAL IN THE YEAR BEFORE START-UP:
C
C  THE WORKING CAPITAL IS TAKEN TO BE 20% OF THE THIRD
C  OPERATING YEAR'S OPERATING COSTS. THIS IS TO AVOID THE CASE
C  WHERE MINE PRIMARY DEVELOPMENT JUST CARRIES OVER INTO THE EARLY
C  PART OF THE MINING PERIOD.
C
      WC = 0.2 * (COST(5,NYEAR(6,1)+2) + COST(8,NYEAR(6,1)+2))
      CFINV(NYEAR(6,1)-1) = CFINV(NYEAR(6,1)-1) + WC
C
C  ADD ENVIRONMENTAL IMPACT STATEMENT COST:
C
      DO 215 J = IQA, IQS
215   CFINV(J) = CFINV(J) + COST(13,J)
C
C  (B) ADDITION OF PLANT & EQUIPMENT COST IN THE OPERATING
C  YEARS:
C
      DO 220 J = IQY, IQI
      CFINV(J) = COST(6,J)
220   CONTINUE
C
C  (C) ADDITION OF RECLAMATION AND BACKFILLING FOR THE OPEN PIT
C  CASE IN THE YEARS AFTER MINING CEASES:
C
      IF (FLAG(2) .LT. 0.0) GO TO 231
      N = NYEAR(6,2) + 1
      DO 230 J = N, IQE
      CFINV(J) = COST(11,J) + COST(12,J)
230   CONTINUE
C
C  IF OTHER INCOME IS AVAILABLE, SUBTRACT THE DEPRECIATION AND
C  INVESTMENT TAX CREDIT FOR PLANT & EQUIPMENT IN THE YEARS BEFORE
C  START-UP:
```

```
C
231   IF ((FLAG(3) .GT. O.).OR.(MPEQYR(1) .GE. NYEAR(6,1))) GD TD 300
      N = NYEAR(6,1) - 1
      IQU=MPEQYR(1)
      DD 240 J = IQU, N
240   CFINV(J) = CFINV(J) - DEPREC(J) * PARAM(7) - TAXCR(J)
C
C
C  CALCULATIDN DF THE 'DEFERRED DEVELOPMENT WRITE-OFF' (DEFDWD)
C  IN THE CASE WHERE INTANGIBLE DRILLING AND DEVELOPMENT CDST WERE
C  NDT EXPENSED AGAINST OTHER CDMPANY INCDME IN THE YEARS BEFDRE
C  START-UP. THE DEFERRED DEVELDPMENT WRITE-DFF AMDRTIZES
C  THESE INTANGIBLE EXPENDITURES DVER THE DPERATING LIFE DF THE
C  MINE:
C
300   DEFDWO = O.O
      IF (FLAG(3) .LT. O.O) GD TD 400
C
      N = NYEAR(6,1) - 1
      IQG=NYEAR(4,1)
      DO 310 J = IQG, N
      DEFDWO = DEFDWO + CDST(4,J) + CDST(5,J) + DEPREC(J)
310   CONTINUE
      DEFDWO = DEFDWD / MLIFE
C
C
C  CALCULATIDN DF THE EFFECTS DF THE INVESTMENT PERIDD CDSTS DN
C  THE DEPLETION ALLDWANCE:
C
C  CALCULATE TDTAL COST DEPLETIDN ALLOWANCE (COSDPN):
C
400   COSDPN = O.
      N = NYEAR(3,2)
      IF (NYEAR(2,2) .GT. NYEAR(3,2)) N = NYEAR(2,2)
C
      DD 410 J = 1,N
      CDSDPN = CDSDPN + COST(2,J)
      IF (FLAG(3) .LT. O.) GD TD 410
      COSDPN = CDSDPN + CDST(1,J) + CDST(3,J)
410   CDNTINUE
C
C  ADD THE CDST DEPLETABLE ITEMS DF YEARS -2 TD O IF KLEMINIC'S TIME
C  PERIODS ARE BEING USED (IN YEARS -1 AND -2 COST ESCALATIDN IS NDT
C  ADJUSTED FDR):
      IF (FLAG(6) .EQ. O.) GD TD 420
      CDSDPN = COSDPN + YRCDST(2) * 3.
      IF (FLAG(3) .LT. O.) GD TO 420
      CDSDPN = CDSDPN + YRCOST(1) * 3. + YRCDST(3)
```

```
C
420   EXPCS = O.
      IF ((FLAG(7) .GT. O.) .OR. (FLAG(3) .GT. O.)) GO TO 500
C
C  SUM THE EXPENSED EXPLORATION COSTS (EXPCS) FOR SUBTRACTION FROM
C  THE DEPLETION ALLOWANCE:
C
      DO 440 J = 1,N
440   EXPCS = EXPCS + COST(1,J) + COST(3,J)
C
      IF (FLAG(6) .EQ. O.) GO TO 450
C
      EXPCS = EXPCS + YRCOST(1) * 3. + YRCOST(3)
450   EXPCS = EXPCS / EXTAX
C
C
500   CALL NETPV (AVGRD, CFINV, COSDPN, COST, DEFDWO, DEPREC, DTREQ,
     1EXPCS, FLAG, LASTYR, MLIFE, NDT, NYEAR, PARAM, RECMIL,
     2REVESC, ROR, TAXCR, TPD, PRICE)
C
      RETURN
C
      END
C  SUBROUTINE NETPV
C  ----------------
C
C
C  A DO-LOOP WITHIN THIS SUBROUTINE VARIES THE ROR. IT PASSES A
C  STARTING PRICE FOR URANIUM OXIDE TO THE SUBROUTINE 'INCOME'
C  WHICH RETURNS THE PRESENT VALUE OF THE RESULTANT CASH FLOWS. THE
C  PRESENT VALUE OF THE INVESTMENT CASH FLOWS IS SUBTRACTED FROM THIS
C  TO GIVE THE NET PRESENT VALUE (NPV). ITERATION AND FINAL INTER-
C  POLATION TAKES PLACE TO DERIVE THE PRICE WHICH GIVES NPV = O.
C  THIS PRICE IS THE REQUIRED URANIUM OXIDE PRICE, IN YEAR C DOLLARS,
C  NECESSARY FOR THE PROJECT TO BE VIABLE.
C
C
      SUBROUTINE NETPV (AVGRD, CFINV, COSDPN, COST, DEFDWO, DEPREC,
     1DTREQ, EXPCS, FLAG, LASTYR, MLIFE, NDT, NYEAR, PARAM,
     2RECMIL, REVESC, ROR, TAXCR, TPD, PRICE)
C
      DIMENSION CFINV(35), COST(13,35), DEPREC(35), DTREQ(40),
     1NYEAR(10,2), PARAM(12), PRICE(5,40),
     2REVESC(35), ROR(6), TAXCR(35)
      INTEGER FLAG(7)
C
      REAL NPV1,NPV2
C  THIS IS THE STARTING POINT OF THE DO-LOOP WHICH VARIES THE ROR:
```

```
C
      OO 300 I = 1,6
      IF (ROR(I) .LT. 0.0) GO TO 400
      DCF = ROR(I)
C
C  PRESENT VALUE THE INVESTMENT CASH FLOWS:
C
      PVINV = 0.0
      OO 10 J = 1, LASTYR
      PVINV = PVINV + CFINV(J) / ((1. + OCF)**J)
10    CONTINUE
C
C  OECISION AS TO THE REQUIREMENT FOR A NEW STARTING PRICE
C  CALCULATION:
C
C  FOR ROR(1), IF THE CHANGE IN O/T IS WITHIN THE LIMITS OF 0.9 ANO
C  1.2 TIMES THE PREVIOUS O/T, PRICE(1,NOT-1) IS USEO AS THE STARTING
C  PRICE ON WHICH TO BASE THE PRICE INCREMENTS. FOR THE OTHER ROR'S,
C  IF THE OIFFERENCE BETWEEN ROR(I) AND ROR(I-1) IS POSITIVE ANO LESS
C  THAN 5%, THE PREVIOUS PRICE IS USEO AS THE STARTING PRICE.
C
      IF (1 .EQ. 1) GO TO 20
      OIFF = (ROR(I) - ROR(I-1)) * 100.
      IF ((OIFF.LE.5.0) .AND. (OIFF .GT. 0.0)) GO TO 30
C
20    IF(NOT.LE.1) GO TO 21
      IF((I .EQ. 1) .ANO. (NDT .GT. 1) .AND. (OTREQ(NDT) /
     1OTREQ(NDT-1).LE.1.2) .ANO. (DTREQ(NOT) / DTREQ(NDT-1)
     2.GE. 0.9)) GO TO 40
C
C  CALCULATION OF A NEW STARTING PRICE FOR URANIUM OXIOE IN $/LB:
C
C  THIS INVOLVES CALCULATING THE ANNUAL INCOME CASH FLOWS REQUIREO
C  TO GIVE NPV = O WHEN PRESENT VALUEO TO MATCH THE PRESENT VALUE
C  OF THE INVESTMENT CASH FLOWS. FROM THESE ANNUAL CASH FLOWS ANO
C  USING SIMPLIFYING ASSUMPTIONS, AN APPROXIMATION OF THE PRICE REQUIRED
C  TO GENERATE THE INCOME CASH FLOWS IS CALCULATEO.
C
C  FUTURE VALUE PVINV TO THE BEGINNING OF THE MINE START-UP YEAR:
C
21    FV = PVINV *.(1. + OCF) ** (NYEAR(6,1) - 1)
C
C  DETERMINATION OF THE ANNUAL INCOME CASH FLOW REQUIREMENT (AF):
C
      AF = FV * (OCF * (1. + OCF)**MLIFE) / (((1. + OCF)**MLIFE) -1.)
C
C  CALCULATION OF THE NECESSARY ANNUAL GROSS REVENUE (GREV):
C
```

```
C  THIS IS DONE FOR THE THIRD OPERATING YEAR. THERE IS ASSUMED TO BE
C  NO ROYALTY PAYMENT AND THE DEPLETION ALLOWANCE TO BE 50% OF TAXABLE
C  INCOME BEFORE DEPLETION IS TAKEN. 'COSTOP' IS THE OPERATING COST AND
C  INCLUDES MINE PRIMARY DEVELOPMENT.
C
      N = NYEAR(6,1) + 2
      COSTOP = COST(8,N) + COST(5,N)
      GREV = (AF - DEPREC(N) - DEFDWO) * (1. - 0.5 * PARAM(7))
     1+COSTOP + DEPREC(N) + DEFDWO
C
C  'SPRICE' IS THE STARTING PRICE IN YEAR O DOLLARS:
      SPRICE = GREV / (2000. * RECMIL * AVGRD * TPD * 365.
     1* REVESC(N))
      GO TO 100
C
30    SPRICE = PRICE(I-1,NDT)
      GO TO 100
C
40    SPRICE = PRICE(1,NDT-1)
C
C
100   CALL INCOME (AVGRD, CFINV, COSDPN, COST, DCF, DEFDWO, DEPREC,
     1EXPCS, FLAG, MLIFE, NYEAR, PARAM, RECMIL, REVESC,
     2SPRICE, TAXCR, TPD, PVCF)
C
      NPV1 = O.
      NPV2 = PVCF - PVINV
C
C  DETERMINE WHETHER TO USE POSITIVE OR NEGATIVE PRICE INCREMENTS,
C  OR WHETHER TO ACCEPT THE PRICE AS IS.
C
      IF (NPV2) 200, 240, 220
C
C  POSITIVE INCREMENTS USED:
C
200   SPRICE = SPRICE + PARAM(12)
C
      CALL INCOME (AVGRD, CFINV, COSDPN, COST, DCF, DEFDWO, DEPREC,
     1EXPCS, FLAG, MLIFE, NYEAR, PARAM, RECMIL, REVESC,
     2SPRICE, TAXCR, TPD, PVCF)
C
      NPV1 = NPV2
      NPV2 = PVCF - PVINV
      IF (NPV2) 200, 240, 210
C
C  INTERPOLATE:
C
210   SPRICE = SPRICE - (NPV2 / (NPV2 - NPV1)) * PARAM(12)
```

```
      GO TO 240
C
C  NEGATIVE INCREMENTS USED:
C
220   SPRICE = SPRICE - PARAM(12)
      CALL INCOME (AVGRO, CFINV, COSDPN, COST, DCF, DEFOWD, DEPREC,
     1EXPCS, FLAG, MLIFE, NYEAR, PARAM, RECMIL, REVESC,
     2SPRICE, TAXCR, TPD, PVCF)
C
      NPV1 = NPV2
      NPV2 = PVCF - PVINV
C
      IF (NPV2) 230, 240, 220
C
C  INTERPOLATE:
C
230   SPRICE = SPRICE + (NPV2 / (NPV2 - NPV1)) * PARAM(12)
      GO TO 240
C
240   PRICE(I,NDT) = SPRICE
C
C  TEST IF THE MAXIMUM PRICE TO BE CALCULATED HAS BEEN EQUALLED
C  OR EXCEEDED FOR ROR(1):
C
      IF (PRICE(1,NDT) .GE. PARAM(11)) GO TO 400
C
300   CONTINUE
C
400   RETURN
C
      END
C  SUBROUTINE INCOME
C  -----------------
C
C
C  THIS SUBROUTINE CALCULATES THE ANNUAL INCOME CASH FLOWS
C  DERIVED FROM THE URANIUM OXIDE PRICE TRANSFERRED FROM THE
C  SUBROUTINE 'NETPV'. IT THEN PRESENT VALUES THESE TO YEAR O AND
C  RETURNS THIS VALUE TO THE SUBROUTINE 'NETPV'.
C
C
      SUBROUTINE INCOME (AVGRO, CFINV, COSDPN, COST, DCF, DEFOWD,
     1DEPREC, EXPCS, FLAG, MLIFE, NYEAR, PARAM, RECMIL,
     2REVESC, SPRICE, TAXCR, TPD, PVCF)
C
      DIMENSION CFINV(35), COST(13,35), DEPREC(35),
    , 1NYEAR(10,2), PARAM(12), REVESC(35), TAXCR(35)
      INTEGER FLAG(7)
```

```
C
      REAL LOSSF, LOSSF1
C
      PVCF = 0.
C
C  DECIDE WHETHER ONLY THE SIMPLIFIED METHOD IS REQUIRED FOR THE
C  UNDERGROUND CASE:
C
      IF (FLAG(7) .GT. 0.) GO TO 400
      CSDPN = COSDPN
      EXPLCS = EXPCS
      KOUNT = -1
      LOSSF1 = 0.
      SUMTAX = 0.
      TAXCRF = 0.
C
C  THIS METHOD CALCULATES EACH YEAR'S CASH FLOWS INDIVIDUALLY:
C
      IQY=NYEAR(6,1)
      IQI=NYEAR(6,2)
10    DO 200 J = IQY, IQI
C
      KOUNT = KOUNT + 1
      DEPL = 0.
C
C  LOSS CARRIED FORWARD:
C
      LOSSF = LOSSF1
C
C  DETERMINE THE ANNUAL GROSS REVENUE:
C
      GREV = SPRICE * REVESC(J) * 365. * 2000.0 * TPD * RECMIL * AVGRD
C
C  DETERMINE THE ANNUAL ROYALTY BASED ON THE MINE MOUTH ORE VALUE
C  (I.E. GROSS REVENUE LESS HAULAGE AND MILLING COSTS):
C
      ROYAL = PARAM(5) * (GREV - COST(9,J) - COST(10,J))
C
C  GROSS REVENUE AFTER ROYALTY:
C
      GRAR = GREV - ROYAL
C
C  TAXABLE INCOME BEFORE DEPLETION:
C
      TIBDPL = GRAR - COST(8,J) - COST(5,J) - COST(12,J) - DEPREC(J)
     1-DEFDWO - LOSSF
C
C  CALCULATE PERCENTAGE DEPLETION ALLOWANCE:
```

```
C
      IF ((TIBDPL.LE.O.) .OR. (PARAM(6) .EQ. O.)) GO TO 20
      DEPLPC = PARAM(6) * GRAR
C
C  UPPER LIMIT ON PERCENTAGE DEPLETION ALLOWANCE:
C
      DEPL = O.5 * TIBDPL
C
C  SELECT THE SMALLER OF THE ABOVE TWO:
C
      IF (DEPLPC .LT. DEPL) DEPL = DEPLPC
C
C  SELECT THE LARGER OF COST AND PERCENTAGE DEPLETION ALLOWANCE:
20    IF (CSDPN .EQ. O.) GO TO 30
C
      DEPLCS = CSDPN / (MLIFE - KOUNT)
      IF (DEPLCS .GT. DEPL) DEPL = DEPLCS
C
C  FIND THE BALANCE OF COST DEPLETION ALLOWANCE TO BE CARRIED FORWARD:
C
      CSDPN = CSDPN - DEPL
      IF (CSDPN .LT. O.) CSDPN = O.
C
C  RECOVER EXPENSED EXPLORATION COSTS FROM THE DEPLETION ALLOWANCE
C  AND CARRY FORWARD ANY BALANCE:
C
30    IF ((EXPLCS .EQ. O.) .OR. (DEPL .EQ. O.)) GO TO 40
C
      DEPL = DEPL - EXPLCS
      IF (DEPL .GE. O.) EXPLCS = O.
      IF (DEPL .GE. O.) GO TO 40
      EXPLCS = ABS(DEPL)
      DEPL = O.
C
C  TAXABLE INCOME AFTER DEPLETION ALLOWANCE TAKEN:
C
40    TIADPL = TIBDPL - DEPL
C
C  IF NO ESPENSING OF TAX LOSSES AGAINST OTHER INCOME IS POSSIBLE,
C  CALCULATE THE LOS CARRIED FORWARD FOR FUTURE DEDUCTION (LOSSF1):
C
      IF (FLAG(3) .LT. O.) GO TO 50
      IF (TIADPL .GE. O.) LOSSF1 = O.
      IF (TIADPL .GE. O.) GO TO 50
      LOSSF1 = ABS(TIADPL)
      TIADPL = O.
C
C  TAX:
```

```
C
50     TAX = TIADPL * PARAM(7)
C
C  IF OTHER INCOME IS AVAILABLE EACH YEAR'S INVESTMENT TAX CREDIT
C  IS TAKEN IMMEDIATELY. OTHERWISE A MAXIMUM OF 50% OF THE TAX
C  LIABILITY IS TAKEN WITH THE REMAINING INVESTMENT TAX CREDIT
C  BEING CARRIED FORWARD FOR FUTURE DEDUCTION.
C
       IF (PARAM(8) .EQ. O.) GO TO 100
       IF (FLAG(3) .GT. O.) GO TO 60
C
       TAX = TAX - TAXCR(J)
       GO TO 100
C
60     TAXCRF = TAXCR(J) + TAXCRF
       IF ((TAX .EQ. O.) .OR. (TAXCRF .EQ. O.)) GO TO 100
       TAX1 = TAX - TAXCRF
       IF (TAX1 .GE. 0.5 * TAX) GO TO 70
       TAX = 0.5 * TAX
       TAXCRF = TAXCRF - TAX
       GO TO 100
C
70     TAX = TAX1
       TAXCRF = O.
C
C  NET INCOME CASH FLOW:
C
100    CASHFL = TIADPL - LOSSF1 - TAX + DEPREC(J) + DEFDWO + LOSSF
      1+ DEPL
C
C  PRESENT VALUE THE NET INCOME CASH FLOW AND SUM WITH THE
C  PREVIOUS TOTAL:
C
       PVCF = PVCF + CASHFL / ((1.0 + DCF) ** J)
C
C  IF NO EXTERNAL TAXABLE INCOME IS AVAILABLE THEN SUM THE TAX FOR
C  THE FINAL 3 OPERATING YEARS. THIS WILL BE USED FOR TAKING TAX
C  LOSSES AGAINST ON THE BACKFILLING AND RECLAMATION IN THE 3 YEARS
C  AFTER MINING CEASES.
C
       IF ((NYEAR(6,2) - J .GE. 2) .AND. (FLAG(3) .GT. O.) .AND.
      1(FLAG(2) .GT. O.)) SUMTAX = SUMTAX + TAX
C
200    CONTINUE
C
       IF ((FLAG(3) .LT. O.) .OR. (FLAG(2) .LT. O.) .OR. (SUMTAX
      1.EQ. O.)) GO TO 300
C
```

```
C  RECOVER THE ABOVE MENTIONED TAX LOSS FROM BACKFILLING AND RECLA-
C  MATION UP TO THE VALUE OF 'SUMTAX'. PRESENT VALUE THAT RECOVERED.
C
      M = NYEAR(6,2) + 1
      N = NYEAR(6,2) + 3
C
      DO 210 J = M,N
      TXLOSS = CFINV(J) * PARAM(7)
      SUMTX = SUMTAX - TXLOSS
      IF (SUMTX .LT. 0.) SUMTX = 0.
      TXGAIN = SUMTAX - SUMTX
      PVCF = PVCF + TXGAIN / (1. + DCF) ** J
      IF (SUMTX .EQ. 0.) GO TO 300
      SUMTAX = SUMTX
210   CONTINUE
C
300   RETURN
C
400   CONTINUE
C  THE FOLLOWING SIMPLIFIED CALCULATION FOR THE UNDERGROUND CASE
C  ASSUMES CONSTANT INCOME CASH FLOWS FOR THE ENTIRE MINE OPERATING
C  LIFE (I.E. BOTH CONSTANT REVENUES AND COSTS):
C
      GREV = SPRICE * 365. * 2000.0 * TPD * RECMIL * AVGRD
      J= NYEAR(6,1)
      ROYAL = PARAM(5) * (GREV - COST(9,J) - COST(10,J))
      GRAR = GREV - ROYAL
      TIBDPL = GRAR - COST(8,J) - DEPREC(J) - DEFDWO
      IF (PARAM(6).EQ. 0.) GO TO 410
      DEPLPC = PARAM(6) * GRAR
      DEPL = 0.5 * TIBDPL
      IF (DEPL .LT. 0.) DEPL = 0.
      IF (DEPLPC .LT. DEPL) DEPL = DEPLPC
410   CSDPN = COSDPN / MLIFE
      IF (CSDPN .GT. DEPL) DEPL = CSDPN
      TIADPL = TIBDPL - DEPL
C
C  ANNUAL NET INCOME CASH FLOWS:
C
      ANCF = TIADPL * (1. - PARAM(7)) + DEPREC(J) + DEFDWO + DEPL
C
C  PRESENT VALUE THESE CASH FLOWS FROM OVER THE MINE LIFE
C  TO THE BEGINNING OF THE START-UP YEAR:
C
      PVANCF=ANCF*((1.+DCF)**MLIFE-1.)/(DCF*(1.+DCF)**MLIFE)
C
C  PRESENT VALUE PVANCF TO YEAR 0:
C
```

```
      PVCF = PVANCF / (1. + DCF) ** (NYEAR(6,1) - 1)
C
      RETURN
C
      END
```